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UNIVERSITY OF ILLINOIS BULLETIN

ISSUED WEEKLY

Vol. XIII

AUGUST 7, 1916

No. 49

[Entered as second-class matter Dec. 11, 1912, at the Post Office at Urbana, Ill., under the Act of Aug. 24, 1912.]

SUBSIDENCE RESULTING FROM MINING

BY

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AND

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ILLINOIS COAL MINING INVESTIGATIONS COOPERATIVE AGREEMENT

(This report was prepared under a Cooperative Agreement between the Engineering Experiment Station of the University of Illinois, the Illinois State Geological Survey and the U. S. Bureau of Mines.)



BULLETIN No. 91

ENGINEERING EXPERIMENT STATION

PUBLISHED BY THE UNIVERSITY OF ILLINOIS, URBANA

CHAPMAN AND HALL, LTD., LONDON
EUROPEAN AGENT

THE Engineering Experiment Station was established by act of the Board of Trustees, December 8, 1903. It is the purpose of the Station to carry on investigations along various lines of engineering and to study problems of importance to professional engineers and to the manufacturing, railway, mining, constructional, and industrial interests of the State.

The control of the Engineering Experiment Station is vested in the heads of the several departments of the College of Engineering. These constitute the Station Staff and, with the Director, determine the character of the investigations to be undertaken. The work is carried on under the supervision of the Staff, sometimes by research fellows as graduate work, sometimes by members of the instructional staff of the College of Engineering, but more frequently by investigators belonging to the Station corps.

The volume and number at the top of the title page of the cover are merely arbitrary numbers and refer to the general publications of the University of Illinois; *either above the title or below the seal is given the number of Engineering Experiment Station bulletin or circular, which should be used in referring to these publications.*

The present bulletin is issued under a cooperative agreement between the Engineering Experiment Station of the University of Illinois, the State Geological Survey and United States Bureau of Mines. The reports of this cooperative investigation are issued in the form of bulletins by the Engineering Experiment Station, State Geological Survey and the United States Bureau of Mines. For bulletins issued by the Engineering Experiment Station, address Engineering Experiment Station, Urbana, Illinois; for those issued by the State Geological Survey, address State Geological Survey, Urbana, Illinois; and for those issued by the United States Bureau of Mines, address the Director, United States Bureau of Mines, Washington, D. C.

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and

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SUBSIDENCE RESULTING FROM MINING

INTRODUCTION.

The subject of the subsidence of the earth's crust as a result of underground excavation due to mining has attracted widespread attention for some years past, but particularly during recent years. With the extension of mining and the increased value of the surface above the mines in many localities, the growth of towns in mining regions, and the extension of railroads over mining properties, the subject is one that will be of increasing interest as time goes on, not alone to those engaged in mining coal and ores, but to the railroads, municipalities, and other owners and users of the surface that may be affected by mining operations.

That the subject is not one of mere local interest is shown by the widespread distribution of surface subsidence as described in the following pages.

This bulletin has been prepared not with a view of bringing forward any new theories in regard to the subject, but it is in the nature of a reconnaissance and a statement of present knowledge of the subject, based upon the literature available up to the present time. It and a companion preliminary Cooperation bulletin on subsidence in Illinois by Dr. Young, which will be issued by the Illinois Geological Survey, are intended as studies upon which to base a detailed cooperative investigation of subsidence conditions in Illinois.

This bulletin represents the result of a study of the literature on the subject. Much of the text is an abstract of this literature, supplemented by extensive private correspondence by the writers, and by a study of conditions in western Pennsylvania, in West Virginia, and in Maryland as given by office data and by an intimate acquaintance with the subsidence problem in the anthracite fields of Pennsylvania.

The authors are particularly indebted to the several anthracite subsidence commissions for the use of unpublished reports, and to a number of engineers for private data, for some of which it has been possible to give due credit in the text, while other data of a confidential nature has had to be incorporated without due credit.

This preliminary study of subsidence literature and of the conditions in Illinois suggests the advisability of undertaking a detailed study of the problem in Illinois. This study may extend over a number of years

in the future. To begin such a study, several groups of mines should be selected; one group in northern Illinois, one in the central part of the state, and another in the southern part. At each mine selected monuments should be erected, and the elevation of these monuments taken at intervals. In connection with these surface observations the conditions in the mine should be noted as closely as possible, in the hope that gradually data will be collected upon which it will be possible to determine the probabilities of and the extent of subsidence upon the surface when underground conditions are known, to determine the size of pillars which will most effectively prevent loss of coal due to squeezes and will properly protect a given surface area.

For the preparation of the extensive bibliography, for the preparation of the abstracts of literature, and for the detailed presentation of the data collected, credit is due entirely to Dr. Young, the undersigned having acted mainly by assisting in the gathering of data and in an advisory capacity in the preparation of the manuscript.

H. H. STOEK.

CHAPTER I.

NATURE AND EXTENT OF SUBSIDENCE PROBLEM

The removal of solid minerals from the earth's crust produces cavities, and thus the equilibrium which has previously existed is disturbed. If the cavities caused by the mining operations are not of great extent, or, even if long, are narrow, this disturbance may be apparent only as a local movement and may cause only occasional falls of rock from the roof. If the excavation is wide as well as long, the unsupported strata above the excavation will tend to sag under their own weight and, if their texture will not permit the bending movement necessary for the strata to become adapted to the new conditions, cracks and fissures resulting in extensive falls of roof will occur. Successively, the overlying beds may break and fall until the disturbance extends to the surface.

If the overlying measures bend without breaking and sag until finally they are supported by the floor of the excavation, the strata at greater height may sag successively and in a corresponding manner. Eventually, this movement may extend to the surface, the disturbance generally being less extensive as the vertical distance from the excavation increases.

In estimating the weight upon any coal seam or other mineral deposit due to the overlying rock, it is customary to assume that this weight is distributed more or less uniformly over the entire deposit. When a portion of a bed of mineral is removed, the burden carried per unit of area by the unmined portion becomes greater than the burden carried before any portion of the deposit was mined, because the weight formerly distributed over the deposit is now concentrated upon the pillars. The extent of the increase of burden on the pillars depends upon the extent of removal of the material of the bed, assuming that the overlying rock does not break in such a way as to relieve the stress on the pillars. If the pillars are not strong enough to support the increased load, or if the underlying bed does not have sufficient bearing power to resist the increased pressure, a movement will begin which is commonly called a "squeeze"* or a "creep." Depending upon the depth of the mining operations and the geological conditions, the "squeeze" may cause an extensive vertical movement which may reach to the surface. The

*The Pennsylvania Mine Cave Commission gave the following definition: "A 'squeeze' is caused by the general subsidence of the strata overlying the coal bed, due to a partial failure of the pillars; when this subsidence radiates from origin it is called a 'creep'." Another meaning of "creep" is movement of the floor, due to pressure of pillars.

removal of coal or other bedded minerals from any considerable area, therefore, at once develops the problem of the support of the surface which involves certain factors requiring careful attention by the mine operator before extensive excavations are made. If the operator, for commercial reasons, meets these problems in a manner that is not in harmony with the prevailing ideas of conservation, a remedy should be sought which will secure for the public the greatest continuing benefit.

Upon the opening of a new mine, the following questions may well be asked:

1. Is the owner of the surface, if other than the owner of the mine or mining rights, legally entitled to surface support?

2. Is the material to be mined at such a depth that mining of all of it will not disturb the surface?

3. If the removal of all the deposit will cause surface subsidence, what percentage of the deposit left in pillars will prevent subsidence?

4. What is the ratio between the value of the material in the pillars necessary to prevent surface subsidence and the value of the surface? What would be the charge per ton against this pillar material if the surface were bought outright?

5. What amount and what extent of subsidence may be expected under the conditions of operation most economical at the time?

6. Upon what basis will it be possible to adjust claims for damages?

7. What will it cost to restore the surface for agricultural uses after all the deposit has been removed?

There are certain questions which the public and the state should answer at an early date:

1. Shall the coal or other mineral now in the ground be brought to the surface and used or shall it be left in the ground, serving like worthless rock, only to support the surface?

2. Assuming that the removal of all the material will temporarily prevent the use of that part of the surface overlying the area being mined, will it be better policy for the state to see to it that all the merchantable material is mined and then have the surface restored, or will it be wiser to permit nearly one-half the material to be lost permanently in the effort to avoid temporary injury to the surface?

3. If the mine operator is required by law to protect the surface, shall anything be done to prevent his leaving a large percentage of the deposit in the ground, never to be recovered and simply to support the surface?

Scientists who have investigated the national resources have em-

phasized the fact that the supply of minerals is not inexhaustible and that at the present rate of increase in production the exhaustion of the supply of some of the most important ones is not far off, as time is measured in the life of a nation. In the case of coal, one of the means by which the life of our supply may be extended is by recovering all, or at least a much greater percentage than is recovered at present, of the coal in the ground. If the extent of the entire area underlaid with workable coal beds be compared with the extent of tillable land not underlaid with coal, it will be noted that the actual area that might be affected by surface subsidence is relatively small. When it is realized that land affected by subsidence may in most cases be restored to service for agriculture after all the deposit has been removed, it may be rightly urged that the mine operators remove much more of the coal than is taken under present conditions, when preservation of the surface is frequently the determining factor in deciding the amount to be mined. Since mineral once lost by improper mining or left in pillars in abandoned mines is lost forever, the maximum recovery consistent with safe mining is of prime importance and is fundamental. The problems, therefore, are to discover what effect mining under the existing physical conditions will have upon the surface, to anticipate and to reduce to a minimum possible surface subsidence and finally to discover the best means of harmonizing and coordinating the various industrial and commercial interests involved.

As will be noted in the discussion of the legal considerations involved in the problem* the legal rights of the several parties interested in the minerals, in the surface, and in the other forms of property in the community must be considered both relatively and absolutely. A study of the subsidence problem from various angles shows the complexity of "conservation" applied to mining, to agriculture, and to other interests at the same time. The complexity seems to increase when efforts are made to coordinate these various interests.

RECORDS OF DAMAGE TO SURFACE.

While the technical press contains many reports of surface subsidence attributed to mining operations there are in America only a few reliable records, available for study, showing the exact amount of subsidence of the surface after the mineral deposit has been mined. However, there are a number of instances in which European engineers have kept records of surface levels extending through long periods of years.

*See Ch. VI.

Surface movements have in many instances been disastrous; records of damage to property being available both in Europe and in America. In the following section a number of the most important instances of damage to property resulting from mining operations are presented. These instances show that the problem is of widespread interest and is not a local one.

Belgium.

Although serious subsidence adjacent to salt mines was noted in England in 1850* and instances of damage by coal mining are recorded in British technical literature, the problem of surface subsidence due to mining operations seems to have been studied first in Belgium. In the early part of the nineteenth century it was claimed that coal mining about Liege, Belgium, was causing damage to buildings, and in the year 1839 complaints were filed with the city officials on account of damages to property. As a direct result of these complaints, the city appointed a committee to report upon the problem and in filing its report the committee established the necessary restrictions required for the safety of the city and determined the size of adequate safety pillars.

The Belgian engineer, Gonot, formulated a theory of subsidence in 1839 and some years later published a pamphlet dealing with the damage to a row of houses adjacent to the mine of the D'Avroy Bovene Company, claiming the mining company was responsible for the damage done. The mining company published a reply to Gonot in 1858. The Provincial Government appointed two engineers to investigate the cause of the damage to the houses and they reported that the houses were not damaged by coal mining.

By a decree of May 31, 1858, the Minister of Public Works appointed a special committee to report on the influence of mining upon the surface and also to review the rules of the committee appointed in 1839. The committee of 1858 endorsed the recommendations of the committee of 1839.

The disturbance of the surface about Liege continued and G. Dumont was appointed to investigate the matter. In his report† he supported the fundamental principle of Gonot's theory but made certain reservations in its application. He placed the responsibility for the surface disturbances upon the mining companies. The Colliery Owners' Association

*Trans. I. M. E., Vol. 19, p. 249, 1899.

†"Des Affaissements Du Sol Produits par l'Exploitation houillère." Liege, 1871.

published a statement* pointing out the fallibility of Gonot's theory but admitted the applicability of the theory to relatively flat seams.

Since 1875 considerable attention has been given to the problem in Belgium, and the situation has been complicated by the mining of coal from superimposed beds.

England, Scotland, and Wales.

Considerable attention has been given to the subsidence problem in England, Scotland, and Wales, owing to the extent of the coal and salt measures, to the importance of the coal industry, and to the proximity of the mines to centers of population. In the early days of coal mining in Great Britain it was customary to leave pillars, but as mining practice improved a portion of the coal in the pillars was removed. In discussing early methods of working coal, Bulman and Redmayne† refer to surface subsidence resulting from the removal of pillar coal as follows: "The date at which it became customary to remove pillars formed by a previous working has been a point of some importance in determining claims for damage to the surface, and many such claims in which the point arose have led to legal proceedings. That damage of this kind was done at an early date is proved by the records of the Halmote Court for the County of Durham. Early in the fifteenth century there was an inquiry before that court about a case which had occurred in the parish of Whickham in which it is recorded: 'It is found by the jury that John de Penrith is injured by a coal mine of Rogers de Thorton so that the house of the said John is almost thrown down, to the damage of the said John of 200 pounds, assessed by the jury; therefore it is considered that the said Roger repair the said house to the value aforesaid, or satisfy the said sum.'"[‡]

Since the year 1860 a number of British mining engineers and operators have written upon the general subject of subsidence and support of excavations. Subsidence has resulted from salt mining operations, as well as from coal mining, and owing to the nature and extent of the salt deposits the effect upon the surface has been even more disastrous than the effect of coal mining. Salt mining has been carried on in the vicinity of Northwich, Cheshire, for many years. In a depth of 390 feet there is a total thickness of almost 200 feet of salt in four beds, the thinnest being 5 feet thick and two being each approximately 90 feet. The shallowest bed is covered by 32 feet of soil and by 92 feet of salt

*"Des Affaissements Du Sol Attribués a l'Exploitation houillère." Liege, 1875.

†Bulman, H. F., and Redmayne, R. A. S. "Colliery Working and Management," p. 9.

‡"History of Durham." Francis Whellan & Co.

marl. Shafts were sunk to the upper bed shortly after 1670 and the pillar-and-room system was used. Pillars from 12 to 21 feet square were left to support the surface, but these pillars were weakened in time by the dissolving action of the water which seeped through the roof and was pumped out as brine. Surface breaks occurred which were generally marked by brine pools. In 1750 the first serious breaks occurred near the main street of Northwich. These old breaks have been filled and buildings have been erected directly over them. Since 1750 numerous breaks have occurred throughout the salt district.* After 1781 all new shafts were sunk to the second bed, which is nearly 92 feet thick and is separated from the upper bed by about 28 feet of hard marl. The most serious subsidence occurred in 1880, and the locality is now covered by a lake about 30 acres in area and of considerable depth.

The drilling of brine wells has increased the rapidity with which the pillars have become weakened and has hastened subsidence in the vicinity of the old mines. Brine streams or channels have been formed underground between the wells and old shafts, and subsidence is greatest near these underground streams.

Owing to the seriousness of the subsidence over an area of 600 acres, frame buildings are used, as these may be blocked up and restored after the most serious surface movement has abated.

The local Board of Trade was asked by the Salt Chamber of Commerce in 1871 to have a report made upon the local situation. This request was referred by the Board of Trade to the Secretary of State, who directed Mr. Joseph Dickinson, Inspector of Mines, to make a report. In March, 1873, Mr. Dickinson presented to Parliament a report which was published under the heading, "Landslips in the Salt Districts."

In 1881 there was introduced in Parliament a bill which proposed to give relief to the owners of damaged property in the salt district. This bill failed to pass, but in 1891 a bill was passed providing for Compensation Boards to be formed in the salt districts. These boards were empowered to levy a tax, not exceeding 3d. on every 1,000 gallons of brine pumped, to form a fund to compensate owners of property damaged. This act was put into force, and in 1896 a provision was added, limiting such compensation to private holders of property and excluding all local authorities, gas and water companies, railway and canal companies, and all pumpers of brine, no compensation whatever being allowed them if any of their property were injured by subsidence.

*Ward, T. "Subsidence In and Around the Town of Northwich in Cheshire." *Trans. Inst. Min. Engrs.*, Vol. 19, p. 241, 1889-1890.

The examples of surface subsidence due to coal mining in England and Wales are very numerous. Much agricultural land has been damaged and also various improvements, including buildings, railroads, bridges, railroad tunnels, canals, reservoirs, and streets and highways.

In one district the Great Western Railway had to fill 60,000 to 70,000 cubic yards annually. A canal in South Staffordshire has been raised 20 feet. Coal mining under the Merthyr tunnel, $1\frac{1}{2}$ miles long, produced a total subsidence of 10 feet in part of the tunnel. The settlement throughout the length of the tunnel was not uniform and part of the line had to be cut down to make the grade uniform.*

In the South Staffordshire district there has been considerable difficulty in securing suitable reservoir sites owing to the fact that coal mining has extended under most of the land and owing also to the fact that the bed is thick and nearly all the coal has been removed. A 3,500,000-gallon reservoir was built on a site which had been undermined thirty-four years before. When the reservoir was filled, the subsidence amounted to $1\frac{1}{4}$ to 2 inches. The cracks were filled with cement and the reservoir has since given no trouble. Another 43,000,000-gallon reservoir was constructed in the coal district in which there are three workable beds of coal, one being 8 feet 3 inches thick and lying at a depth of 1,200 feet, while 66 feet below it is a 6-foot 3-inch seam, and 80 feet above it is a 6-foot seam. These have been worked and one end of the reservoir has lowered 4 feet more than the other, the great difference in elevation between the ends being attributed to a fault.

In order to reduce the damage to water mains the practice in the English mining districts is to use lead joints instead of the "turned and bored" pipes.†

In the Midland and South Yorkshire coal fields the measures overlying the coal are principally shales, and mining at a depth as great as 2,000 feet has caused some subsidence.‡

In his address as President of the Institution of Mining Engineers, W. T. Lewis called attention to the seriousness of subsidence in Wales, stating that the surface sinks from 10 to 15 feet on account of mining at 1,800 to 2,400 feet.§ The removal of 4 feet 9 inches of coal and underclay, constituting a shaft pillar, at a depth of 2,108 feet is reported to have caused surface subsidence of 3 feet 6 inches at the South Kirby Col-

*Ingles, J. C. "Subsidence Due to Coal Workings." *Proceedings, Inst. of Civ. Engineers*, Vol. 135, p. 131, 1898.

†*Proc. Inst. of Civil Engrs.*, Vol. 135, p. 156, 1898.

‡*Eng. and Min. Jour.*, Vol. 84, p. 196, 1907.

§*Trans. Inst. Min. Engrs.*, Vol. 22, p. 290, 1901.

liery.* This amount of subsidence is unusual in the district and was attributed to the crushing of a shaft pillar in the overlying Barnsley bed. A maximum subsidence of 1.74 feet resulted from longwall mining of 5 feet of coal at a depth of 1,595 feet in Derbyshire.† Mr. James Barrow cited an instance of mining coal 5 feet 6 inches to 6 feet 6 inches at a depth of 2,400 feet. The longwall method was used and the debris resulting from the working of the seam and the brushing of the roof was stowed underground.‡ Subsidence caused buildings on the surface to crack, water and gas mains to be broken, and bridges to be squeezed and distorted. Mr. J. Kirkup reported that the mining by longwall of a seam 22 inches thick produced cracks in walls and caused damage in pipes in workings in a seam 279 feet above. Moreover, a careful survey showed that the movement in the upper seam extended in advance of the workings in the lower seam.§ Mr. I. T. Rees has reported on subsidence resulting from longwall mining in the coal field in South Wales. The lower seam worked was from 3 to 4 feet thick and was well stowed. "Three hundred sixty feet above this seam, workings had been prosecuted in another seam in advance of the seam below, and although there were 360 feet of intervening strata, and the openings caused by working the seams were well stowed, yet the workings of the seam above were affected a distance of 150 feet in advance of the workings of the seam below."§ In 1912 the Wearmouth Coal Company, Ltd. (Sunderland), was forced to stop working the Hulton seam, which employed 400 men, on account of the heavy charges for surface damage resulting from subsidence. In one case the charge was \$500,000.**

France.

In France subsidence has been noted in the salt mining district as well as in the coal fields. In French-Lorraine, the salt measures extend under an area approximately 9 by 19 miles. The thickness of the beds varies from 33 to 230 feet and the beds lie at a depth of 300 feet or more. The salt has been removed in part by solution methods, which produce large chambers, and, owing to the great size of these chambers and to the character of the roof, extensive falls of roof rock have occurred. The subsidence has generally taken place slowly, but where the

*Snow, Charles "Removal of a Shaft Pillar at South Kirby Colliery." Trans. Inst. Min. Engrs., Vol. 46, p. 8, 1913.

†Hay, W. "Damage to Surface Buildings Caused by Underground Workings." Trans. Inst. Min. Engrs., Vol. 34, p. 427, 1908.

‡Proc. South Wales Inst. of Engrs., Vol. 20, p. 356, 1897.

§Kirkup, J. Discussion of paper on "The Absolute Roof of Mines." Trans. Inst. Min. Engrs., Vol. 31, p. 180, 1905.

§Proc. South Wales Inst. of Engrs., Vol. 20, p. 359, 1897.

**Trans. Inst. Min. Eng., Vol. 44, p. 533, 1912.

covering is limestone there have been sudden breaks which have caused extensive damage. Among the serious surface movements reported are one in 1879 at St. Nicholas and one at Ars-sur-Meurthe in 1876.*

Fayol made a number of observations of subsidence at the Commentry Mines, as well as laboratory experiments, and published the results of his observations, including the levels taken at these mines from August, 1879, to May, 1885.† He advanced a theory of subsidence which was essentially different from that formulated by the Belgian engineer, Gonot.

Germany.

Probably the first important German publication on surface subsidence in connection with mining was by A. Schulz, in 1867.‡ He investigated the dimensions of safety pillars and the angle of break in the Saarbruck field. The problem was considered so important that in 1868 the Prussian government appointed a commission of four engineers to investigate the effect upon the surface of mining operations in the coal fields of Belgium, England, France, and Rhenish Prussia. In 1869 von Dechen wrote upon the subsidence in and about the city of Essen. He had previously (in 1866) emphasized the importance of studying the part played by the heavy marl beds overlying the coal measures.

In 1867, von Sparre contributed a paper upon the "afterbreak."§ In 1894, the project of a canal between Herne and Ruhrort aroused a discussion in regard to the stability of the surface over which the proposed canal was to run. The Board of Mines of Dortmund conducted observations in the Dortmund district and the results were published in 1897.§

In the Dortmund district there have been a number of accidents** due to thrust movements, and in the Ruhr coal fields miniature earthquakes, supposed to have been due to coal mining, have caused considerable damage.

Methods of reducing surface subsidence by hydraulic stowing have received much attention from the mining operators of Upper Silesia and Westfalia.

The coal-beds under Zwickau, Saxony, are situated at a depth of

*Bailly, L. "Subsidence Due to Salt Workings in French-Lorraine." *Annales des Mines*, Ser. 10, Vol. 5, pp. 403-494, 1904.

†Bul. de la Société de l'Industrie Minérale, II^e série, Vol. 14, p. 818, 1885.

‡Zeit. für B., H., u. S.-W., 1867.

§Glückauf, 1867.

§Zeit. für B., H., u. S.-W., p. 372, 1897.

**Zeit. für B., H., u. S.-W., Vol. 51, p. 439, 1903.

600 to 2,500 feet. Beginning in 1885, observations were made at eighty-two points to determine the surface movement resulting from mining operations. After twelve years it was noted that subsidence amounted to 85.2 inches, due to mining at 600 to 900 feet. At 1,500 feet, the subsidence was only 9.17 inches.

The town of Eisleben in the Mansfeld mining district was seriously damaged by earth shocks, fissures, and subsidence during the years from 1892 to 1896. Various theories were advanced concerning the cause of these disturbances. Some held that they were due to the dissolving of various salt deposits by underground water thus producing caverns, and that as these caverns became of great extent, large falls of overlying rock caused the shocks and the subsidence. Others held that in addition to the solution of the salt, carbonated waters were leaching the deeper lying dolomitic formations, and when these became honeycombed, they were unable to support the load concentrated on the natural pillars resulting from the solution of part of the overlying salt beds. At the Mansfeld Copper Mine copper bearing shale from 12 to 20 inches thick was being mined by a longwall method at a depth of from 900 to 1,800 feet. Public opinion blamed the mining company and, as a result of arbitration, the company paid \$125,000 damages.*

Potash mining at Stassfurt in beds 50 feet thick and dipping 40 degrees has caused serious subsidence. Stone buildings have sunk as much as 20 feet, rows of houses have been removed to firm ground, and chimneys and towers are standing 5 degrees from the vertical.†

On June 10, 1910, surface subsidence, described as a local earthquake, occurred at the Consolidation Mine. "The part of the coal measures most affected formed part of an undulation or 'saddle.' The forces at work were of such intensity, and so irregular in their action, that steel rails were twisted into corkscrew like shapes, and in a section of the saddle 10 feet in length, two lines of rails, water-pipes, signal-wires, and rope-way were found crushed together into a bundle of about 12 to 16 inches thick."‡

Austria.

The review of the theories of subsidence presented by Austrian engineers, as given by Goldreich,§ indicates that as early as 1859 there

*Lang, Otto "Subsidences at Eisleben." Bul. de la Societe Belge de Geologie, Vol. 11, p. 190, 1898.

†Private correspondence.

‡Zeit. f. B., H., u. S.-W., Vol. 59, p. 68, 1911. Abstracted in Trans. Inst. Min. Eng., Vol. 41, p. 587, 1910.

§Goldreich, A. H. "Die Theorie der Bodensenkungen in Kohlengebieten." Berlin, 1913.

were regulations controlling the mining of coal under railways in Austria. Director W. Jicinsky published a treatise on "The Subsidences and Breaks of the Surface in Consequence of Coal Mining."* The publication by Rziha† was the first contribution by an Austrian engineer to the theories of subsidence. Most of the Austrian writings on subsidence have been on the problems of the Ostrau-Karwin coal district. Goldreich has studied the problem principally through years of observation in railway engineering.

One of the most serious disasters in Austria resulting from surface subsidence occurred July 19, 1895, at Brux, Bohemia,‡ where the brown-coal seams lie nearly horizontal at a depth of 325 feet, covered by clay-shale interspersed with quicksand from 10 to 65 feet thick. There are in all four seams having an average total thickness of 80 feet. Some filling had been used, but sand broke into the mine, and it is estimated that two million cubic feet of sand entered the workings. Numerous holes were formed on the surface, rendering sixty-six houses uninhabitable and making 2,000 people homeless.

Another serious disaster occurred at Raibi, Bohemia, at a lead mine, where an attempt was being made to secure an adequate water supply through underground workings. Two short drifts were being driven through the rock toward water-bearing sands, and though a borehole was kept ahead of each drift, a blast so weakened the cover that the roof broke and a rush of sand followed. A large hole was made on the surface and without warning a small municipal hospital dropped forty feet, causing the death of seven of the inmates.§

India.

Coal mining in the Bengal field has caused disturbance of the surface along the outcrop. At the Khoira Colliery the mining of 10 feet 6 inches of coal dipping 30 degrees has caused complete subsidence of the surface where the workings are shallow. At the Barrea Colliery, owing to the value of the rice land, stowing has been used to reduce the amount of subsidence.§ In the same field mining of thick coal overlaid by thick beds of sandstone has been attended by extensive falls of roof which have produced fatal air-blasts.**

*Oestrr. Zeit. für B., u. H.-W., p. 457, 1876.

†Oestrr. Zeit. für B., u. H.-W., 1882.

‡Helmhacker, R. "Land Subsidence at Brux, Bohemia." Trans. Inst. Min. Engrs., Vol. 10, p. 583, 1895-96.

§Oestrr. Zeit. für B., u. H.-W., Vol. 58, p. 31, 1910.

§Stonier, G. A. "Bengal Coal Fields." Trans. Inst. Min. Engrs., Vol. 28, p. 537, 1904.

**Adamson, T. Trans. Inst. Min. Engrs., Vol. 29, p. 425, 1905.

South Africa.

In the diamond mines of South Africa there have been rushes of mud into the mines. These have been due to water softening the steep walls of the open pits which then give way and fill the mine openings.*

In connection with the Rand gold mines there has been surface subsidence similar to that caused by deep longwall coal mining.† The maximum depth from which mining has affected the surface has been 710 feet on the Champ d'Or. Other depths from which subsidence has extended to the surface are 566 feet at the Bonanza mine, 650 feet at the May Consolidated, 480 feet at the Treasury, 340 feet at the New Kleinfontein, 490 feet at the New Heriot, and 425 feet at the Windsor. At the Gueldenhuis Deep an area 1,000 feet on the strike by 620 feet on the dip, at depths of from 650 to 924 feet, with an average stoping width of 15 feet caved suddenly but no sinking of the surface resulted. Similar results at other mines have led South African engineers to conclude that cavings of stopes below 1,000 feet in depth will not affect the surface.

UNITED STATES.

Alabama.

In Alabama little attention has been given to the subsidence problem, owing to the fact that many of the coal mining companies have been operating under land to which they hold the title and of which the surface has relatively little value in comparison with the coal. At least 90 per cent of the mining is at a depth of less than 400 feet. Some cracks have extended to the surface and when damage has been caused to property not owned by the mining companies, it has usually been possible for the mining companies to make settlements not greatly out of proportion to the damage done.‡ At the present time some mining is being carried on where the cover is as much as from 800 to 1,200 feet and very little trouble is being experienced.

Idaho.

In the metal mines of the Coeur d'Alene district, disturbances have been noted which apparently are due to causes similar to those

*Williams, G. F. "Diamond Mines of South Africa," pp 400-404.

†Richardson, A. "Subsidence in Underground Mines," Jour. of the Chem. Met. and Min. Soc. of S. A., Mar., 1907; Eng. and Min. Jour., Vol. 84, p. 196, 1907.

‡Private correspondence.

which have produced air-blasts in other ore mines.* Recently at an Idaho ore mine two men were killed and four seriously injured by an air-blast.

Illinois.

Subsidence of the surface due to coal mining has attracted attention in Illinois for a number of years. In the early days of coal mining, when only the shallow beds were mined, the surface was seriously damaged, but in those days the price of farm lands was low and most of the mining was conducted in sections not thickly populated. The first important suit for damages that was appealed to the higher courts in Illinois was in Sangamon county in 1880 (Wilms vs. Jess, 94 Ill. 464). Since that date but few subsidence cases have been tried in the higher courts in Illinois, most of the claims for damages being settled by arbitration or by decisions of the lower court.

An investigation in 1914 showed that there had been surface subsidence in the most important coal producing counties of the state. Twenty-four of the total of fifty-two counties in which coal is mined produced 94 per cent of the coal mined in the state in 1913, and in twenty-one of these counties, subsidence, in various stages and degrees of intensity, was noted.

The reported damages include injury to farm lands and buildings, to city buildings and streets, to railroads and highways, and to domestic and municipal water supplies. Large tracts of farm land in northern Illinois are reported to be damaged by disturbance of surface drainage due to subsidence. There has been litigation to determine the extent to which mining is responsible for the inundation of lands adjacent to waterways and streams.

Few instances of injury to persons by subsidence of the surface have been reported. Mining at shallow depths has permitted the movement of large bodies of surficial material, at time resulting in a rush of sand, clay, and water into the mine, causing serious damage to the mine. Fortunately there have been but few such instances of personal injury to miners from such rushes. This may be due largely to the precautionary steps which have been taken since the accident in the long-wall field in 1883 at Braidwood. The disaster at Mine No. 2 of the Diamond Coal Company near Braidwood, Illinois, was due to the inrush of water through surface breaks caused by subsidence.†

*Sizer, F. L. "An Air-Blast or Earth Movement." *Mines and Minerals*, Vol. 33, p. 87, 1912.

†Coal Report of Illinois, p. 97, 1883. Roy, A. "History of the Coal Mines of the U. S." Columbus, pp. 190-194, 3d Ed., 1907.

A horizontal bed, 3 feet thick, was being mined at a depth of about 100 feet. The overlying strata are largely shale and clay. Longwall mining had permitted the surface to sink and, at various points at which the rock cover was thin, cracks and breaks extended for some distance up into the surficial material. In February, 1883, there had been a heavy fall of snow followed by a thaw and rain. On February 15, vast sheets of water were standing on the prairie and on the following day a number of the miners did not go to work, as they feared that the water would break through into the mine. At 11 a. m. on February 16 there occurred a cave which permitted a great inrush of water from the surface. The flow of water cut out a larger inlet to the mine and in a short time all of the low points on the roadways were filled with water so that escape was impossible. In three hours the entire mine was filled and the water rose to the surface. Sixty-one men and boys failed to escape before the mine was flooded.

A comprehensive report upon subsidence in Illinois has been prepared by L. E. Young and will appear as a contribution of the Cooperative Investigation by the Illinois Geological Survey.

Indiana.

The following data regarding subsidence in Indiana have been furnished by H. I. Smith, mining engineer, U. S. Bureau of Mines:

A few squeezes have been reported in the mines near Evansville. At one mine operated under the Ohio River at a depth of 260 feet below the river bottom no trouble from the overlying river was reported and at another mine operated at a depth of about 300 feet no loss of coal due to squeezes was reported when about 55 per cent of the coal was removed. Local squeezes occurred but were stopped by a barrier pillar and the coal was reached from the next set of parallel entries.

Probably the greatest damage from subsidence in Indiana has been in Clay county over the upper and lower block coal beds. In one instance in which there was from 20 to 40 feet of cover, consisting of shale with 2 to 6 feet of clay and soil on top, the overlying material was so yielding that an outline of each pillar or stump could be traced on the surface. After a period of twenty years these sinks are said to have evened up, leaving little or no trace upon the surface. Over recent workings succeeding rooms can be traced on the surface by pit-holes or sinks. In some cases the strata have broken through to the surface and the depth of the hole is the same as the thickness of the coal, that is, about five feet. Local residents state that within five years farm land again be-

comes tillable and in twenty years the depressions have disappeared. This does not apply to large swags which cover a number of adjacent rooms and which in some cases must be drained. A good example of these swags is found at West Seeleyville in the field north of the inter-urban stop and between the interurban stop and the Vandalia Railroad. These swags are said to have occurred one year before the mine was abandoned, and the coal is said to lie at a depth of 110 feet.

In Vigo county, about $1\frac{1}{2}$ miles north of Miami, a number of small sinks were observed in a cultivated field. On the opposite side of the road, in Clay county, one of the sinks has broken through and is about five feet deep. Other depressions were observed in the line of these workings, but were not broken through.

A private correspondent reports that in Linton, about six years ago, one side of a concrete block house dropped from $2\frac{1}{2}$ to 3 feet. The break extended from top to bottom and passed through the blocks instead of following the joints. Two or three years ago one section of an L-shaped school house was badly damaged and the front end of a store fell in. The court records show five or six suits at Linton for recovery of damages due to mining. The coal is worked on the room-and-pillar system.

Kansas.

Longwall mining at shallow depths in Kansas in the vicinity of Osage City has caused some subsidence, but no damage has been done to sidewalks, brick buildings, etc. The coal is from 12 to 18 inches thick, and lies at a depth of 70 to 80 feet.* Above the coal is a light limestone and upon it rest the upper Pennsylvania strata of alternating shale and limestone.†

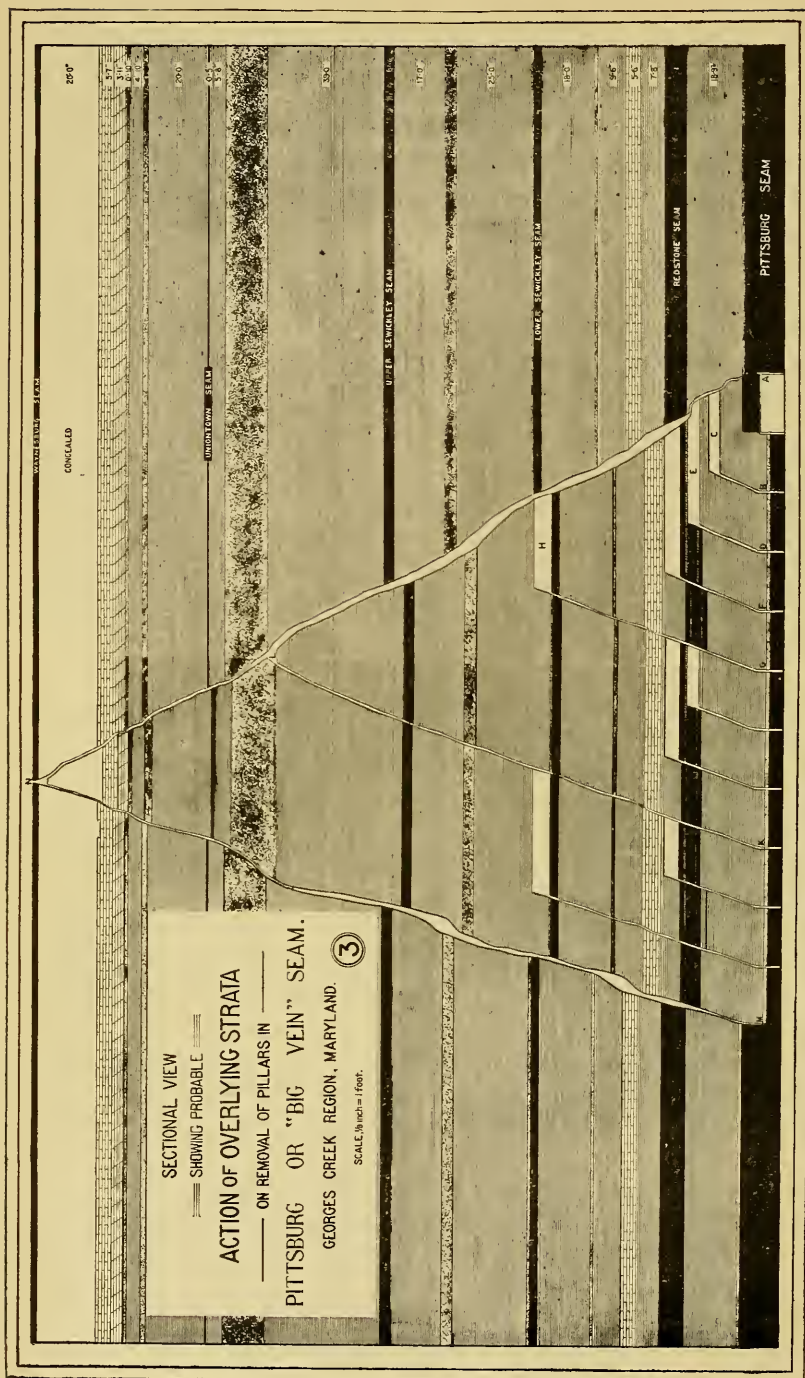
It is reported that subsidence of surface has resulted from the removal of salt by brine-pumping. The salt measures are about 400 feet thick and are covered by a total thickness of 600 feet of beds of shale, limestone, and sandstone.

In the southeastern part of the state mining is conducted on the room-and-pillar system in coal dipping gently toward the west. There have been subsidences, especially near the outcrop, but no extensive damage has been done.

Near Leavenworth longwall mining is carried on at a depth of about 700 feet in a bed 19 to 24 inches thick. There are no published records of subsidence. The surface is rolling and no damage would be likely

*Private correspondence.

†Kansas State Geol. Survey, Vol. 3, p. 70.



except to buildings, paving, pipes or sewers. The numerous stone and brick buildings of the State Penitentiary at Lansing have been undermined by the State Mine, but show no evidences of subsidence. Mines extended under the Missouri River show no seepage of river water.

Maryland.

In the George's Creek Region, Maryland, the coal seam varies from 6 feet 6 inches to 9 feet 10 inches in thickness and, when the pillar coal is removed, falls occur which extend to the surface.* Fig. 1 indicates the supposed effect of the removal of the pillars under the overlying strata where the surface is 250 feet above the coal seam. Subsidence extends

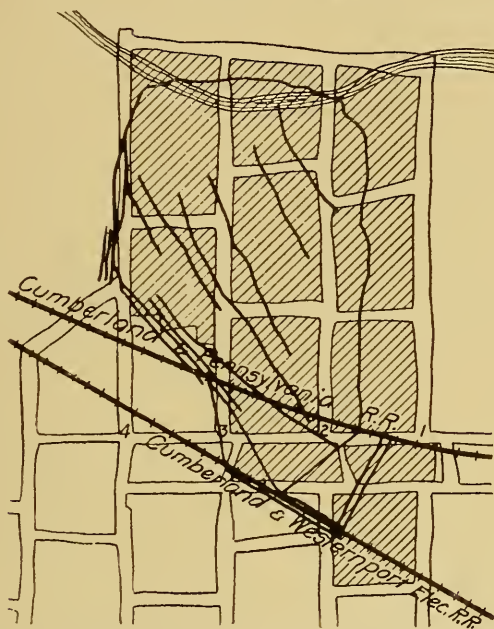


FIG. 2. RELATION OF SURFACE CRACKS TO UNDERGROUND WORKINGS.

to the surface in such a case after the pillars have been drawn back 220 feet. After the first break occurs at *B* due to drawing the pillar at *A*, the entire block of roof sinks and causes the cavity *C*. The distance from *A* to *B* horizontally is 40 feet, or 40 feet of pillar have been taken out when the first fall occurs. The second fall occurs at *D* and the fracture line extends to the space *E* at the Redstone seam. This break occurs

*Reppert, A. E. "Pillar Falls and the Economical Recovering of Coal from Pillars." Proc. West. Va. Coal Mining Institute, p. 110, 1911.

after 60 feet of pillar has been taken out and the break extends almost 40 feet above the floor of the Pittsburgh seam. The third fall extends to *F* and the fourth to *G*, the line of fracture in the latter case extending to the space *H* in the Lower Sewickley seam, 85 feet above the floor of the Pittsburgh seam, the total length of pillars drawn up to this stage being 100 feet. The next break occurs when pillars have been drawn for 160 feet to *K* and the break extends to *L*. When the pillars are drawn back a distance of 220 feet to *M* the fracture extends to the surface at *N*, a height of 250 feet above the bottom of the Pittsburgh seam. This fracture line is approximately correct as shown in Fig. 1 and is based on actual survey and observation of a large number of surface breaks in relation to the mine workings. Fig. 2 indicates the position of surface breaks due to the removal of pillars at a depth of 170 feet over an area 300 feet by 350 feet, the thickness of coal averaging 8 feet. The first surface break occurred between rooms No. 1 and 2 and was about 70 feet from the barrier pillar. The average angle of fracture from the vertical is $22^{\circ} 30'$. The break along the barrier pillar at the top of the rooms was at an angle of 14° from the vertical, while the break along the left hand pillar of No. 4 room was nearly vertical. The conclusion has been drawn that "until a pillar fall extends to the surface, the fracture is conical in shape, but as the pillar line extends down the rooms beyond the first surface break, the strata fracture on a nearly vertical line."*

Michigan.

Copper Mines.—In the deep copper mines of northern Michigan extensive falls of roof have produced air-blasts. At great depths the pillars left to support the roof, or at times masses of poor rock left unmined, show the effect of the tremendous weight upon them. The edges of these pillars fail first and large slabs may burst off and fly some distance. The pillars fail suddenly and the fall of rock may be extensive enough to cause a jar that will be felt on the surface.†

Beginning in 1904 there were a number of caves at the Atlantic mine. The stopes averaged 15 feet in width and extended for a mile along the strike of the lode and for one-half mile down the dip. These falls became so extensive and the pressure on the pillars became so great that the shafts were ruined and the mine was put out of commission, May, 1906.‡

*Op. Cit., p. 113.

†McNair, F. W. "Deep Mining in the Lake Superior District." Eng. and Min. Jour., Vol. 83, p. 322.

‡Stevens, H. J. Copper Handbook, Vol. 10, p. 377.

Extensive falls of the hanging-wall or roof have caused trouble at the Quincy mine. Stopping averages 20 feet in width and in parts of the mine has been carried on to a depth of more than 6,000 feet on the dip of the lode. The sudden and violent compression of air by falls in the mine has caused damage to the levels and shafts and has produced miniature earthquakes.* The general manager of the Quincy mine, in his report for the year 1914, writes as follows:

"On March 25 air-blasts occurred throughout the mine and continued intermittently for a week or ten days. As a consequence, various cross-cuts and drifts were crushed and closed up. No. 6 shaft timbers were seriously crushed between the 51st and 58th levels, and No. 2 shaft was crushed and closed between the 40th and 50th levels. About 500 feet of the crushed section of No. 2 shaft had to be entirely recovered and retimbered at an expense nearly as great as that of sinking a new shaft. In the remaining portion of the damaged shaft about half of the timbers were replaced.

"Below the 50th level the shaft was not damaged by the air-blasts, though the cross-cuts at the 57th, 64th, 65th and 66th levels were entirely closed, and the levels north were badly crushed.

"In earlier days, when air-blasts were little understood, it was the custom to stope out the lode without reference to the shaft. Going through the upper portions of No. 2 and No. 6 shafts is like going down through open stopes, with practically no pillars left to protect the shafts. It was in the lower part of these sections that the caving and crushing took place with such serious results.

"At the present bottom of the mine, pillars are being left 200 feet on each side of the shaft. The air-blasts have never caused any damage to these sections of the shaft.

"Air-blasts have continued with more or less frequency since July, though they have not damaged or retarded the work to any great extent. In order to meet the air-blasts and prevent as far as possible the damages caused by them, as fast as the mining in each stope is finished, the bottom of the stope along the back of the level is filled with poor rock, constituting what is termed 'rib work.' Experience has taught that these rock-packs are the most effective means yet employed to lessen the damages caused by air-blasts. In order, however, that the highest effectiveness possible may be secured within the limits of profitable mining at greater depth, this rib work should be still further strengthened. It is

*Ibid., Vol. 10, p. 1444.

estimated that the voids in the rock give it a shrinkage of about 20 per cent at the present depth of the mine. In order to lessen this shrinkage and strengthen the rib work, the question of filling the voids with bank sand, stamp sand, or crushed rock is receiving serious attention, inasmuch as provision must be made for better and stronger supports to the back of each level, as fast as stoping is finished. During the year \$57,190 was spent for the recovery of the shafts and levels that were damaged by the air-blasts, and \$21,487 has been expended in the fight toward preventing damages by air-blasts."

These rock movements appear to be confined to the deeper portions of the mines and no effect is noted at the surface except a vibration, giving the effect of an earthquake. No subsidence is reported.

Iron Mines.—At a number of points on the Lake Superior iron ranges, in Michigan and in Minnesota, the mining of extensive bodies of ore close to the surface has caused the subsidence of large areas. A considerable portion of the iron ranges is covered with glacial deposits and when the bedrock is shattered by mining rushes of sand into the mine may follow.

At several important mines large caves have occurred under important railway lines. Owing to the inclination, volume, and extent of the ore body it was thought that it would be more practical to bring the track to grade by filling than to construct a new line entirely outside the subsiding area. The continued development of the ore body and deeper mining have caused the subsidence to continue from year to year so that now the problem of filling has become a very expensive one for the railroad company.

Missouri.

At Lexington, Missouri, the mining of 20 inches of coal at a depth of 160 feet has caused subsidence amounting in places to the full thickness of the coal. No serious damage has resulted.*

In the Joplin district extensive caves of the surface have resulted from the mining of large bodies of zinc ore at shallow depths, but no detailed study of subsidence has been made.

During the year 1915 a number of mills were damaged through the tailings piles falling into the excavations. One cave-in resulted in the death of several men in the mine by drowning and it seems inevitable that there will be many more caves in the district, particularly in the sheet deposits where small pillars are left.

*Private correspondence.

In the Flat River district large areas have been mined at depths up to 700 feet. The beds are practically horizontal and have good roofs. "In some mined-out areas where the pillars have been removed and slabbed the back has come in, extended over an area of from one to three acres. In two such caved areas that have been examined it has been found that in each case a natural arch has been formed and the caved material has nearly filled the opening to the back. The largest of the caves of this type has run up about 100 feet into the back, which leaves about 400 feet of undisturbed formation above it."*

Oklahoma.

Coal mining in Oklahoma has caused surface disturbance at a number of places. On September 4, 1914, a serious squeeze occurred in Mine No. 1 of the Union Coal Company at Adamson, which resulted in the death of thirteen miners, complete loss of the mine, and minor surface damage. The mine was opened on the pillar-and-room system. Fig. 3 shows a cross-section through the two thin seams that were worked. The lower seam is 4 feet thick and 45 feet above it is a seam

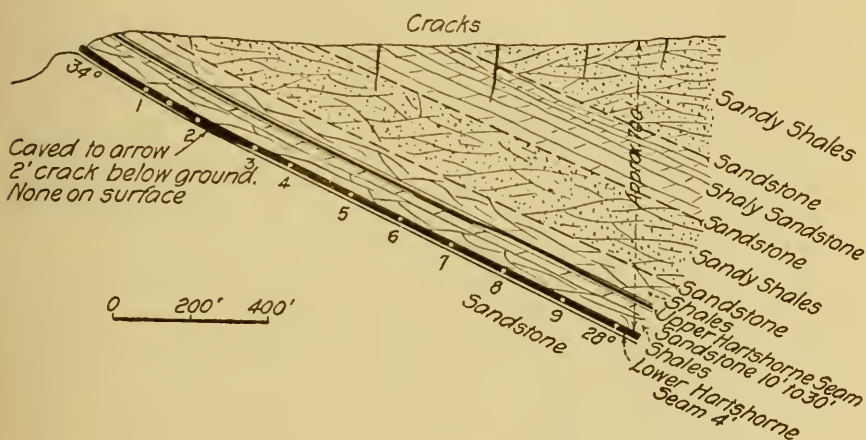


FIG. 3. SECTION THROUGH ADAMSON MINE, OKLAHOMA.

2 feet 3 inches thick, not worked. The beds dip about 30 degrees. Rooms were turned on 33-foot centers, the room pillars being not more than 9 to 12 feet wide and in places much less. The roof was a sandy shale, 28 feet thick, and above it an equal thickness of hard sandstone and a little fireclay. The squeeze came comparatively quickly, completely

*Private correspondence.

closing the slope. A number of cracks appeared on the surface, but no serious surface damage was done.*

There are no records available showing the amount of surface subsidence, but the surface cracks have been located fairly accurately. Assuming that the underground break extended to the tenth level, there was about 700 feet of cover, and the angle of break may be calculated for the cracks farthest from the mouth of the slope. It has been assumed that the large crack over the east side resulted when the 7th East entry was lost. Similarly cracks on the west may be correlated with the underground movement on the 6th West and 9th West entries.

George S. Rice, Chief Mining Engineer, U. S. Bureau of Mines, says in regard to this accident:

"What happened was almost inevitable with a strong roof and increasing depth, where so large a percentage of the coal had been extracted in the advance work and the pillars left standing. Estimating from the map, about 75 per cent of the coal had been taken out by the entries and rooms. As a result, in the lowest level of the mine there was a load of over 3,000 pounds per square inch on the 25 per cent of coal which remained in the entry and room pillars. This is more than bituminous coal can sustain. Therefore, I am inclined to think that the main support of the overlying strata had been carried by arch stresses, the arch being buttressed on one side by the strata near the outcrop and on the other by the dipping strata. Then when the fracture occurred at the latter buttress, it threw the entire weight on the mine pillars, causing them to be crushed. The surface cracks, reported by reliable witnesses to have occurred prior to the collapse, running parallel but in advance (horizontally) of the lowest level, indicated that a shear fracture had occurred in a plane roughly at right angles to the plane of the dipping beds, and when this fracture extended laterally to a sufficient distance, it formed a slip plane which permitted the entire weight of the overlying strata less friction to be thrown upon the pillars, resulting in the collapse of the mine."

Pennsylvania.

Pennsylvania Anthracite Field.—In the United States surface subsidence due to mining operations has received most attention in the anthracite field of Pennsylvania, and notably in the city of Scranton. Of the total area of 176 square miles in the Wyoming region, incorporated boroughs and cities cover 101 square miles.† The city of Scranton ex-

*Brown, G. M. "A Sudden Squeeze in an Oklahoma Mine." *Coal Age*, Vol. 6, p. 533, 1914.

†Report of Pa. State Anthracite Mine Cave Commission, 1913.

tends entirely across the coal field, a distance of five miles, and for the same distance along the valley. Beneath a portion of the city are eleven important coal beds having an average aggregate thickness of 58 feet. It has been estimated that during the seventy-five years of active mining under the city 177,000,000 tons of coal have been produced. This output represents a volume of 198,000,000 cubic yards, an amount in excess of the total estimated excavation in connection with the Panama Canal.* In the early years of mining accurate maps of the mines were not made and preserved and a number of the coal companies made little effort to columnize the pillars in the various coal beds that were worked. These conditions have made it difficult to study the problem and to provide adequate and practical remedies.

While subsidence has occurred in many parts of the city, it was estimated in 1912 that not more than 15 per cent of the area of the city was threatened.

Although surface subsidence had damaged property within the city limits prior to August 29, 1909, the public in general gave the matter little connected attention. This was due largely to the fact that the mining companies hold deeds which permit them to remove the coal without liability for damage to the surface. On the date mentioned, surface subsidence caused serious damage to a school building, which fortunately was not in use at that season of the year.

Following this cave Honorable J. B. Dimmick, then Mayor of Scranton, by approval of the City Council and the Board of Control of the Scranton School District, created a Commission to investigate the physical causes of mine caves and the legal responsibility therefor. The report of this Commission, submitted March 20, 1911, was published in 1912 as Bulletin No. 25, U. S. Bureau of Mines. It was the result largely of the investigations of Eli T. Conner and William Griffith and reviews the existing mining conditions and discusses at length methods for supporting the surface. Considerable attention was given to "flushing" methods of filling and the report contains a chapter by N. H. Darton, entitled "Notes on Sand for Mine Flushing in the Scranton Region." An appendix includes the results of tests to determine the compressive strength of anthracite† and of tests of various kinds of materials for supporting the roof in mine workings.‡

*Bul. 25, U. S. Bureau of Mines, Washington, 1912.

†These tests were made for a committee of the Scranton Engineers' Club in 1900 in the engineering laboratories at Cornell University, Lehigh University, and the Pennsylvania State College.

‡Fifteen tests on the compressive strength of materials for supporting roof were made for this investigation at the Engineering Laboratory of Lehigh University.

The U. S. Bureau of Mines continued investigations along several lines which were discussed in the report by Conner and Griffith and subsequently published the results as bulletins. Bulletin No. 60 contains the investigation by Charles Enzian entitled "Hydraulic Mine Filling; Its Use in the Pennsylvania Anthracite Fields." Bulletin No. 45 by N. H. Darton is a report upon "Sand Available for Filling Mine Workings in the Northern Anthracite Basin of Pennsylvania."

In 1911 Governor Tener appointed the Pennsylvania State Anthracite Mine Cave Commission to investigate the physical conditions and legal rights of surface support. This Commission consisted of:

W. J. Richards, Vice-President and General Manager, Philadelphia and Reading Coal and Iron Co.

G. M. Davies, Mining Contractor.

J. Benjamin Dimmick, Mayor of Scranton.

E. G. Lynett, Editor, Scranton Truth.

W. L. Connell, Coal Operator, Ex-Mayor of Scranton.

R. A. Phillips, General Manager, Coal Department, Delaware, Lackawanna and Western Railroad.

W. H. Lewis, Retired Coal Operator.

Charles Enzian, Mining Engineer, U. S. Bureau of Mines.

W. A. Lathrop, President, Lehigh Coal and Navigation Co.

(Owing to the death of W. A. Lathrop, he was succeeded by S. D. Warriner, President, Lehigh Coal and Navigation Company.)

The investigation of this Commission covered the period from June 12, 1911, to March 1, 1913, when the report was submitted to the Governor and Legislature, the text of the report being printed in the Journal of the Pennsylvania Legislature for 1913, Volume 5, page 5947. This Journal report contains none of the maps or illustrations essential to an understanding of the report, which is available, therefore, only in typewritten form. In addition to the field investigations, the Commission conducted a series of thirty-four tests upon supporting materials, the tests being made at the Bureau of Mines Laboratory in Pittsburgh, in cooperation with the U. S. Bureau of Standards, under the supervision of Charles Enzian.

The 1913 session of the Pennsylvania General Assembly enacted the Davis Mine Cave Law,* which provides for the protection of public highways, streets, etc., and also provides for the creation by municipalities of a Bureau of Mine Inspection and Surface Support, the duties of which are to investigate the mine workings in their relation to the

*Act No. 857. Approved July 26, 1913.

support of public highways. Thus far the City of Scranton is the only municipality in Pennsylvania that has passed an ordinance for the creation of such a bureau. The City of Scranton Bureau of Mine Inspection and Surface Support has been in existence since August 5, 1913, and has investigated a number of mines and made reports upon such investigations. The reports of this Bureau are not in print.

The City Council of Scranton on October 24, 1913, appointed H. D. Johnson and D. T. Williams to prepare a report upon the mine of the Peoples Coal Company, and in their work they were assisted by Chas. Enzian, Mining Engineer of the U. S. Bureau of Mines. The report of these gentlemen, submitted December 12, 1913, was made under the provisions of the Davis Mine Cave Act. It comprises ninety-three typewritten pages and, in addition to reviewing the mining conditions in three city wards, contains much concise information of general application in the study of the problem of surface subsidence.

The subsidence problem in the Pennsylvania anthracite field has been further complicated by the glacial deposits which occasionally are localized in "pot-holes." These pot-holes may extend to a considerable depth below the glacial sheet, making it dangerous to carry on any mining operations near them. When the subsidence of the coal measures extends to a pot-hole filled with sand and water, the water and some of the sand may seep into the mine, and if the subsidence has shattered the intervening strata or if the roof has been thin and weak, a rush of sand may fill the mine workings. The difficulties of mining under such glacial deposits have been recently presented* to the mining profession and fourteen accidents have been noted.

In most of these a large area of the workings has been filled by the rush of glacial material and water, and in several instances extensive surface subsidence resulted.

On June 10, 1914, at the Sugar Notch Mine, a breast in the Kidney bed broke into the wash. The material entering the mine was largely sand and clay in a semi-fluid state, and its volume was estimated at 20,000 cubic yards. This filled several thousand feet of gangways and tunnels, but no lives were lost. The accompanying illustrations show the important data in connection with this accident. Fig. 4 shows the mine workings, the contours of the top of the rock, the location of drill holes, and the important surface features previous to the accident. The cave occurred at the face of breast 15. Fig. 5 shows the conditions

*Bunting, D. "The Limits of Mining Under Heavy Wash." Amer. Inst. of Min. Engineers, Bul. No. 97, p. 1, 1915.

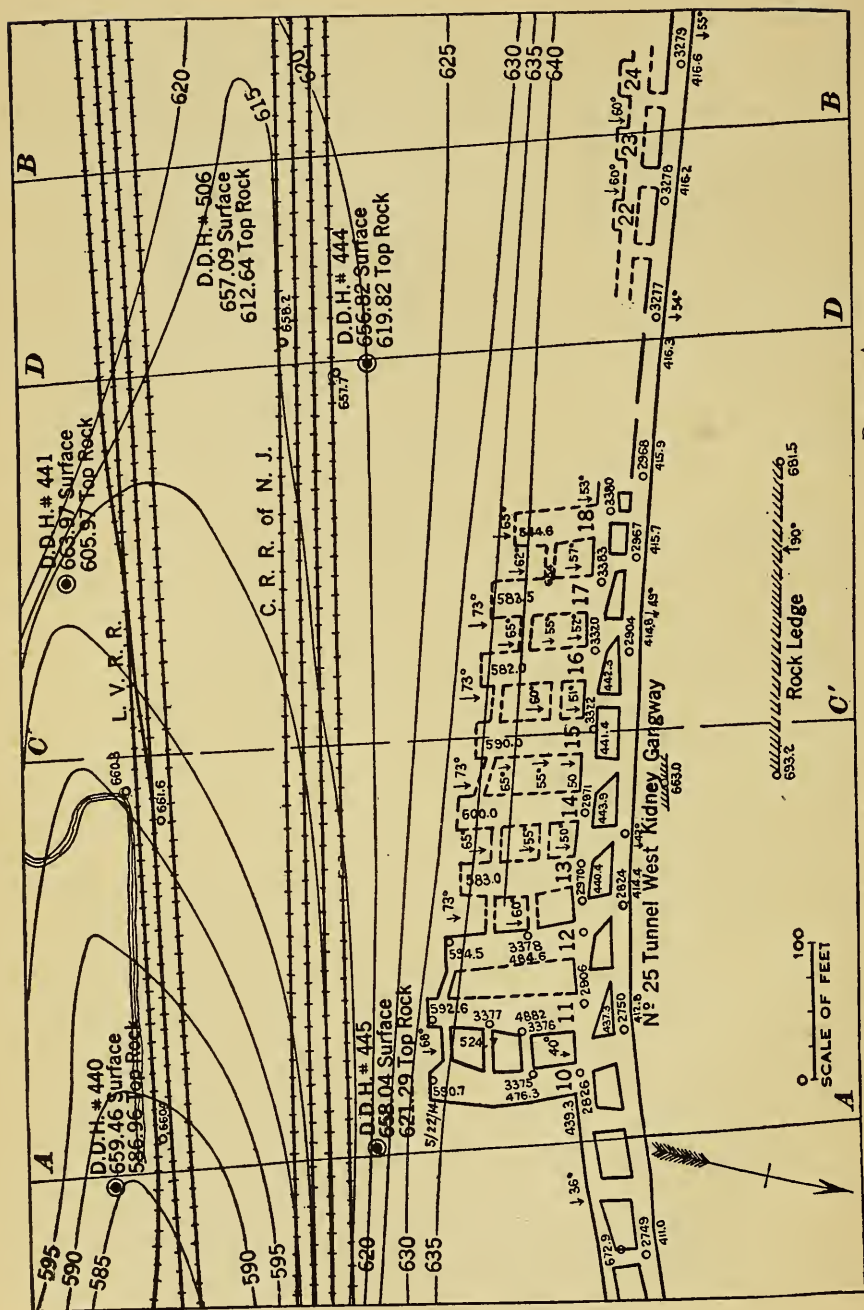


FIG. 4. SUGAR NOTCH MINE; SURFACE AND UNDERGROUND BEFORE ACCIDENT.

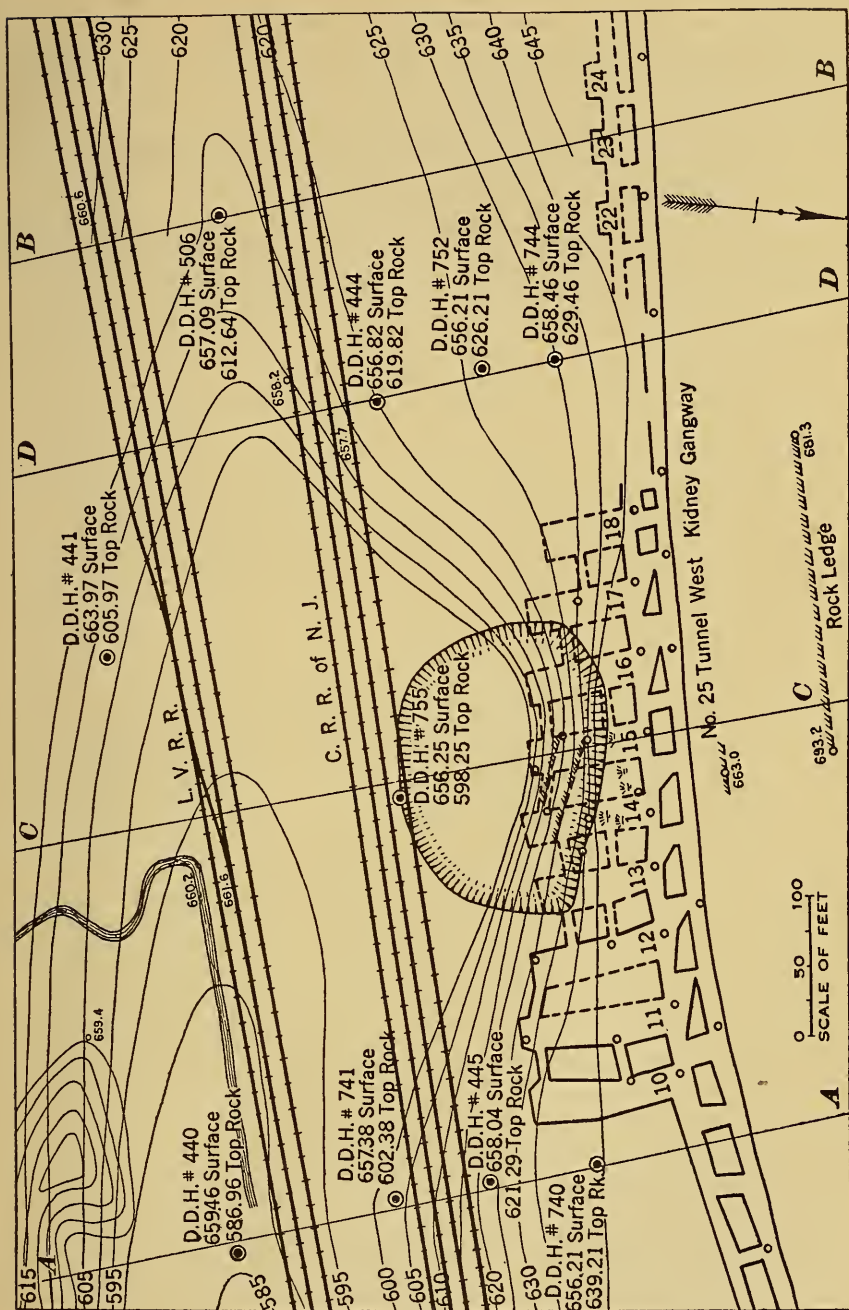


FIG. 5. SURFACE AND UNDERGROUND FEATURES AFTER ACCIDENT.

TABLE 1.

ACCIDENTS IN WYOMING FIELD DUE TO INRUSHES OF SAND AND WATER.

Accident, No.	Date	Mine	Location	Vein	Tapped by	Result
1	July 4, 1872	Burroughs	Plainsville	Hillman	Breast	An inrush of sand and water. A pumpman, the only man in the mine at the time, easily escaped.
2	June 30, 1874	Wanamie No. 18	Wanamie	Red Ash	Breast	Gangway and workings in the vicinity were filled for some distance from the break.
3	Jan., 1882	Maltby	Swoyersville		Rock Plane	Gangways were filled; also the shaft for a vertical height of 90 ft.
4	Apr. 23, 1884	Fuller	Swoyersville	Six Foot	Slope	Slope filled to the top for a distance of 900 ft.
5	, 1884	Ridge	Archbald	Archbald		
6	May, 1885	Ridge	Archbald	Archbald		
7	Dec. 18, 1885	No. 1 Slope	Nanticoke	Ross	Breast	Gangways in the vicinity were completely filled in less than an hour.
8	Aug., 1889	Fuller	Swoyersville		Rock Plane	The planes and all workings tributary to it were filled with sand or water.
9	Mar. 1, 1897	Mt. Look-out	Wyoming	Pittston	Breast	A large area of the workings was filled; no men at work at the time.
10	Dec. 30, 1898	Wanamie No. 18	Wanamie	Copper	Breast	Gangway on lower level filled to a height of 2 ft. for a distance of 300 ft. Depression on the surface 100 ft. east and west and 75 ft. north and south.
11	Feb. 2, 1899	Franklin	Wilkes-Barre	Kidney	Breast	Filled gangway to a height of 3 or 4 ft. for a long distance.
12	Apr. 13, 1899	No. 2 slope	Nanticoke	Hillman	Breast	Gangways were filled for several thousand feet. Breasts had been worked 26 years previous; no men at work in the vicinity. Surface depression was 70 to 80 ft. deep.
13	Apr. 25, 1899	Bliss	Hanover	Hillman	Breast	Gangways and tunnel in the vicinity were filled tight to roof. Conical depression on surface 60 ft. in diameter and 40 ft. deep.
14	June 10, 1914	Sugar Notch No. 9	Sugar Notch	Kidney	Breast	Gangways and tunnels were filled tight to the roof. Depression on surface 150 ft. wide, 210 ft. long and 60 ft. deep.

after the cave and indicates the points where additional drill-holes were put down. Fig. 6 shows the conditions which existed in section C-C, before the accident and indicates the supposed limit of the wash. Fig. 7 shows the conditions after the accident and shows the limits of the wash as proven by the additional drill-holes. The face of breast 15 was at an elevation of +590.0 when the break occurred. The elevation of the surface directly above was +657.0 and it was thought that the bot-

tom of the wash was at +630.0. It was planned to carry the breast to +600.0. The bottom of the wash was actually at +600.0 instead of +630.0.

One of the most serious of the accidents noted was the so-called "Nanticoke disaster" of December 18, 1885, which resulted in the loss of 26 men.* The mine workings tapped a pot-hole which was subsequently found to be over 200 feet deep. The hole was 400 feet away from the present stream and was covered by a culm-bank. The thickness of the rock where the cave occurred has been estimated at from 22

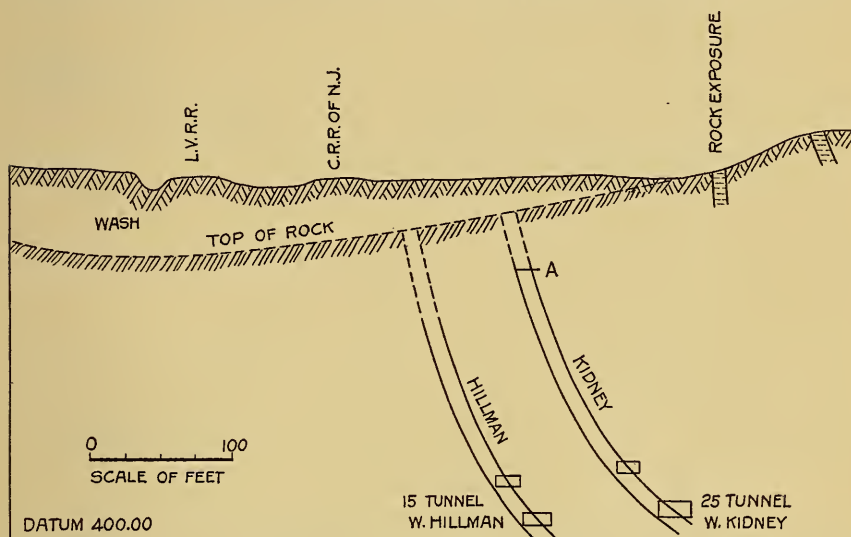


FIG. 6. SUPPOSED CONDITIONS ALONG C-C BEFORE ACCIDENT.

to 48 feet. The subsidence produced a hole in the culm-bank 300 feet across.

One of the most important occurrences of this nature in the anthracite district, as far as amount of material entering the workings is concerned, was the cave at the Prospect colliery of the Lehigh Valley Coal Co., December 12 and 26, 1915.

It was supposed that the rock over the upper bed of coal was 40 to 50 feet thick and that the surface soil was thin, these being the conditions at points near the break. It was found, however, that the rock where the break occurred was only about 10 feet thick and that the

*Williams, G. M. "Dangerous Outcrops," *Mines and Minerals*, Vol. 20, p. 410, 1900; Ashburner, C. A. "The Geologic Relations of the Nanticoke Disaster." *Trans. Amer. Inst. Min. Engrs.*, Vol. 15, p. 629, 1886-87.

remainder of the cover was loose sand, clay and gravel, apparently glacial material deposited in an old valley. Probably the break was due to the collapse of the thin rock at the bottom of a pot-hole.

Two breaks occurred, both at times when a small stream flowing over the loose material was flooded, and a large amount of this material was washed into the mine. It was estimated that about 140,000 cubic

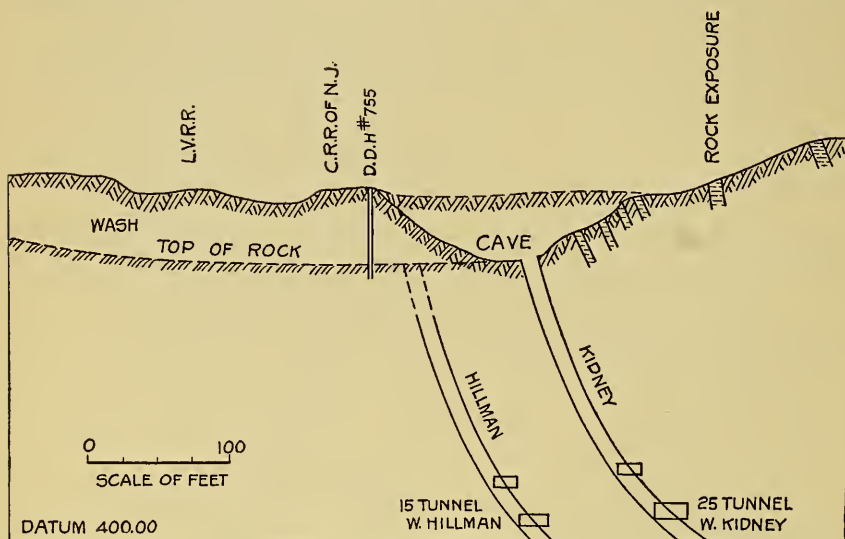


FIG. 7. ACTUAL CONDITIONS ALONG C-C AFTER ACCIDENT.

yards of earth and 350,000,000 gallons of water entered the mine. No lives were lost, but the financial loss was very considerable, as large expense was incurred in changing the channel of the stream, in addition to the cost of pumping out the water and to the loss due to interruption of the work of the colliery.*

Pennsylvania Bituminous Field.—In the bituminous fields of Pennsylvania some damage to the surface has resulted from mining operations, but few references to such subsidence are found in the technical press. Generally the deeds to the coal rights have not required the mining companies to support the surface.

In the Connellsville region the surface is of little value as compared with the coal, the topography is rugged and, although cracks extend through from the mine workings to the surface, little attention is paid to them unless they are near important structures. Mining 8 feet of

*Coal Age, Vol. 9, p. 373, 1916.

coal at a depth of 600 feet produces cracks as much as 20 inches wide. The recovery of coal varies from 84 to 90 per cent. When it is necessary to protect buildings and railroads as much as 25 to 50 per cent of the coal is left in pillars.* It has been found that at shallow depths, up to approximately 150 feet, subsidence will amount to 50 per cent of the thickness of the coal. At greater depths it will be less, approximating 25 per cent at 300 feet. The attempts to correlate data and to generalize from the data available have not been satisfactory. In discussing the observations made in southwestern Pennsylvania, J. P. K. Miller, Chief Engineer of the H. C. Frick Coke Company, said: "The great difference of strata overlying the coal no doubt contributed largely to the great variation noticed throughout the district. In some parts of this region, the stratum immediately above the coal, between it and the sand formation, varies from a few inches to 16 and 20 feet of shale. Where the sandstone is very close to the top of the coal, the subsidence is considerably greater than it is where the shale thickens; then, too, there is a very heavy percentage of limestone and sandstone in the Leisenring district, while immediately southeast of this, or between Uniontown and Fairchance, the sandstone measures rapidly thin out, and this, too, contributes to the variation in subsidence, or, in a word, where the coal has immediately over it a heavy percentage of sandstone measures, the subsidence is greater than where a thick stratum of shales appears immediately above the coal. Of course, this is only our opinion, but it seems to be the only good reason we can give for the difference in subsidence where the cover is approximately the same (thickness). As an illustration, in the territory in the vicinity of Uniontown, where heavy shales appear above the coal, we have observed 18 inches of subsidence where the cover is 300 feet; in the Leisenring district where heavy sandstone measures appear above the coal and there is a thin layer of shale immediately above the coal, the subsidence is approximately 30 inches. There is another condition that, no doubt, contributes largely in bringing about this difference in subsidences and that is the heavy layer of fireclay immediately beneath the seam of coal appearing in the Uniontown district; while very good bottom conditions—'hard bottom'—appear in the Leisenring district, and the writer believes it may be concluded naturally that this difference in the condition of the bottom section has more to do with the difference in subsidence than the first two conditions above mentioned."

*Private correspondence.

The outcrop of the Pittsburgh coal bed extends for many miles in western Pennsylvania, and above these shallow workings many sink-holes have formed. These have attracted very little public attention, as they are considered to be of only a temporary character and most of the buildings above the mined areas are frame and the damage to them has also been only temporary, for if tilted out of line, these buildings have frequently resumed their normal condition after a few months.

Observations made by another company show that the surface subsided 2 feet, 9 inches after practically all of an 8-foot seam had been removed at a depth of 400 feet. The overlying rocks consisted of shales, sandstone and limestone in alternating beds, the thickest limestone bed being 200 feet from the surface and reaching a thickness of 50 feet. There are six other beds of limestone varying from 20 to 30 feet, the total thickness of the seven beds being about 170 feet. The remainder of the column is about equally divided between fireclays, sandstones, and shales. The subsidence took place about twelve months after the mining of the pillars began.*

In the construction of the Greentree Tunnel of the Wabash Pittsburgh Terminal Railway Company in the Pittsburgh district it was found that early mining operations had removed coal from a bed immediately beneath the projected line of the tunnel and that coal had been removed from another bed overlying. W. F. Purdy, Chief Engineer, describes the conditions as follows: "When the heading of the tunnel had proceeded about 500 feet from the west portal we encountered broken ground. The material was fairly solid gray shale which was easy to drill, but none of it could be removed without heavy blasting. At first the only indication of disturbance was that the rock showed soft pockets, and a little later the strata had separated so that large pockets could be excavated without blasting. After having proceeded about 20 feet into the material which had become more or less loosened without our having been able to account satisfactorily for the nature of the ground, the bottom of the heading suddenly broke down about 30 feet back from the face of the heading, permitting partial collapse of the timbering as the settlement was about 2 feet.

"It developed that the broken ground encountered in the heading was at the apex of the mass affected by the subsidence in the mine and the top of the heading was approximately 50 feet above the mine level. Unknown to us at the time, we had been driving the heading for some

*Private correspondence.

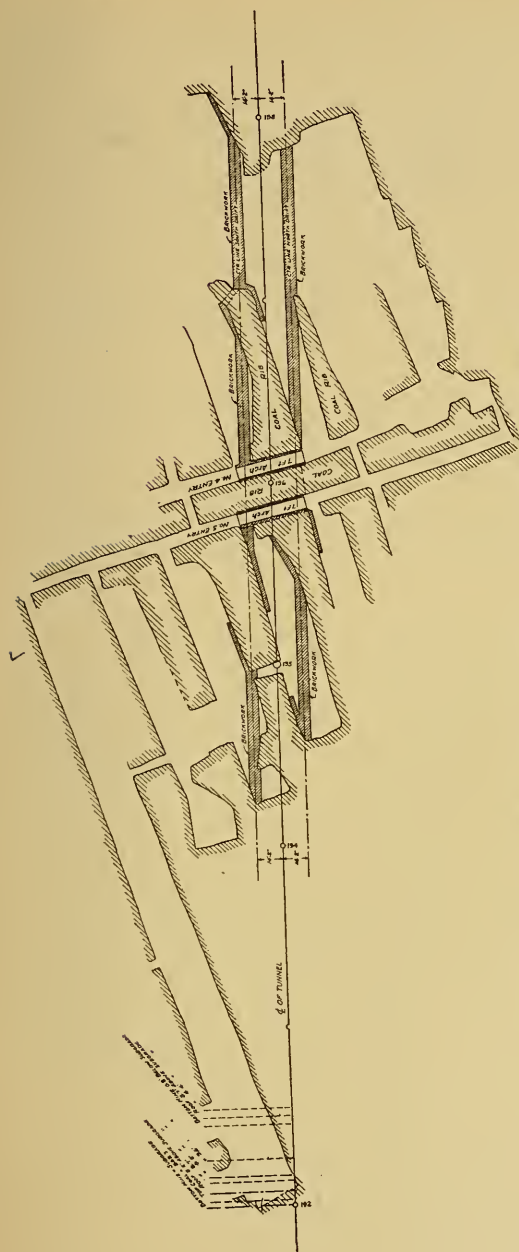


FIG. 8. PLAN OF GREENTREE TUNNEL AND OLD MINE WORKINGS.

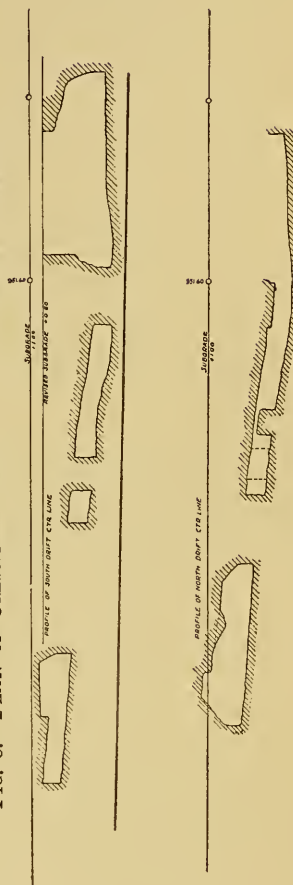


FIG. 9. SECTION THROUGH TUNNEL AND MINE.

distance over the broken ground with a wedge of solid rock between the bottom of the heading and the broken material beneath. When we had reached the point where this continually thinning wedge would no longer support the weight, the ledge broke, practically at right angles to our tunnel, and allowed the timbering to drop as before stated.

"We moved back in the heading to the solid rock and drove a shaft on a steep incline until we reached the floor of the mine about 35 feet below the bottom of our heading. There we found that a considerable area of coal had been mined several years earlier and because of a "swamp" and the apparently heavy expense for pumping the mine was abandoned after having drawn nearly all of the ribs and pillars.

"After reaching the mine level we drove two diverging drifts from the foot of the shaft until we were under the two sides of the tunnel and then carried the two drifts ahead under the prospective tunnel walls. For about 70 feet the ground was entirely broken between the mine and the grade of our tunnel and we built solid brick masonry walls 18 feet in height to provide foundation for the regular tunnel side-walls.

"After leaving the space of about 70 feet already mentioned the ribs and pillars had not been withdrawn and the falling from mine roof was not serious. At some points the roof slate had fallen in to a height about 6 feet above the normal mine roof, and at other places there had been no breakage whatever. We followed up the mine entries and rooms for a distance of about 600 feet at which point on account of the convergence of the tunnel grade toward the mine level the same elevation was common to the floor of mine and the grade of the tunnel.

"The alinement of our tunnel makes an angle of approximately thirty degrees with the rooms in the mine and the work was more complicated on that account. We built solid brick masonry walls diagonally across the mine entries and rooms under each of the two tunnel side-walls. Where there was a space of several feet between the roof of the mine and the normal foundation of tunnel walls, we built the brick walls thick enough to prevent any danger of crushing the slate rock between the brick foundation and the regular tunnel walls. When we reached the point where only two or three feet of natural shale would have been left between the roof of mine and bottom of tunnel wall we cut it all out and carried the brick work up to the grade of the tunnel.

"The tunnel lining is of concrete and the tunnel has been in service for ten years with no sign of settlement or cracking of the concrete.

"We also experienced a good deal of trouble, expense and delay on account of another coal mine above the eastern half of the same tunnel. As the top of our tunnel heading approached the bottom of the coal mine and as the intervening wedge of ground became thinner it became very difficult to support the roof of tunnel and we had much trouble on that account, and also because of the mine drainage which poured through the thin ledge of shale between the roof of tunnel and bottom of mine."

West Virginia.

Owing to the character of the topography and the low value of the surface in the coal districts, very few reliable records are available to show the extent of surface subsidence due to coal mining. Some surface movement has been noted where from 7 to 8 feet of coal has been mined and the pillars drawn at depths over 500 feet. When the thickness of the covers is from 200 to 300 feet, the disturbance is greater and "where the cover is light—from 50 to 150 feet—the cracks are sometimes from 2 to 4 feet in width and show a vertical displacement of from 1 to 2 feet."*

The problem of protecting a seam lying from 70 to 80 feet above the seam now being worked has confronted some of the coal mining companies. It is proposed to mine the upper seam before the pillars are drawn in the lower seam, as the subsidence which follows the mining of the pillars in the lower seam greatly disturbs the overlying seams and makes it unprofitable to mine them.

NATURE OF DAMAGE DUE TO DISTURBANCE OF THE OVERLYING MATERIAL.

The damage resulting from the excavation of minerals may be (a) without the mine or (b) within the mine.

In this study the damage external to the mine is the subject of investigation, the internal damage being noted only when it occurs in connection with external damage.

(a) The damage external to the mine may be due to:

1. The vertical, or horizontal, or both vertical and horizontal movement of surface material or surface structures, caused by the subsidence of the strata overlying the excavation.
2. Surface cracks or fissures due to slips, faults, or shear, or to the tension of the surficial beds.

*Private correspondence.

3. Pit-holes or caves, formed when, instead of gradual and more or less uniform subsidence over a large area, the movement is localized if the excavation is at a shallow depth.
 4. Damage by water on account of the lowering of land below the former drainage channels or the high-water levels, and by the derangement of artificial drainage systems, such as sewers in cities or tiling on farms.
 5. Interference with or destruction of natural or artificial water-supply.
 6. Miniature earthquakes occurring when large masses of rock fall over great areas.
- (b) Within the mine, damage may result from:
1. The inflow of water through cracks or breaks caused by subsidence of the strata.
 2. The inflow of sand through breaks extending to the superficial material.
 3. Local or extensive falls of roof.
 4. The failure of pillars, due to the excessive weight of the superincumbent strata.
 5. "Air-blasts" or "bumps" accompanying the sudden collapse of pillars and the fall of large areas of roof.
 6. Squeezes or creeps.

1. *Nature of Earth Movement.*—Damage to structures on the surface may be the result of either vertical or horizontal movement, or both, and engineering observations in Europe and in America show many interesting facts regarding the extent, the rate, and the duration of surface movements.

"Draw" or "pull" is the variation from the vertical of the line of fracture of rocks that break when the supporting bed or stratum is removed; in other words, the variation from the vertical of the boundary between the disturbed and undisturbed strata. In some cases this is a well-defined plane; in others a zone of indefinite extent. In the case of brittle rocks, the break will be sharp; while in the case of more yielding deposits, such as shales and loose soil, it may be impossible to determine exactly the limits of disturbance.

Several instances of the lifting of objects on the surface have been reported, but no data are available at this time to prove definitely that either a temporary or a permanent elevation of the surface has occurred.

It is claimed that at Northwich, England, where subsidence has resulted from the removal of salt, an elevation of certain streets has occurred.* In Nottinghamshire, England, where a coal seam 3 feet 8 inches thick was mined at a depth of 1,680 feet, it is claimed that there was locally and temporarily an elevation amounting to 4 inches.† Also, it is reported that an "earth tide" is evident in the diamond fields of South Africa, there being noted a rise and fall amounting to 3 inches a day, but this could have no connection with subsidence.‡

The engineer's records at the Warrior Run Colliery of the Lehigh Valley Coal Company in Pennsylvania show that there was a lateral

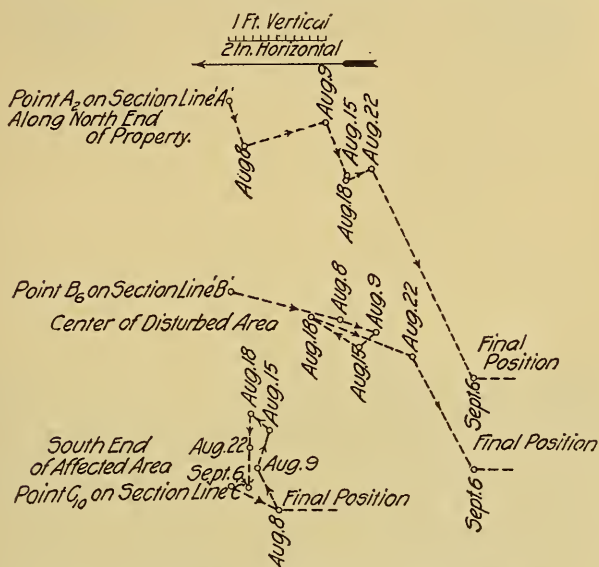


FIG. 10. LATERAL MOVEMENT OF MONUMENTS.

movement of surface monuments as well as vertical movement. (Fig. 10 shows the dates of observation and the amount of the movement.) This resulted from a squeeze in workings extending from 500 to 1,000 feet in depth on a coal seam dipping approximately 30 degrees.¶

Sags of the surface, or depressions without important breaks or cracks, occur when the movement is due to the bending rather than to the breaking of the strata and when the surficial material, without sud-

*Trans. Inst. Min. Engrs., Vol. 19, 241, 1899.

†Proc. Inst. of Civ. Engrs., Vol. 135, p. 114, 1898.

‡Jour. Chem. Met. and Min. Soc. of S. A., October, 1911.

¶Enzian, Charles "The Warrior Run Mine Disaster," Mines and Minerals, Vol. 27, p. 439, 1907.

den movement, accommodates itself to the new inclination of the bed-rock. Observations in the coal districts indicate that the extent and the gradient of such sags are influenced by the rate of advance of the working face, particularly in longwall mining; by the character and

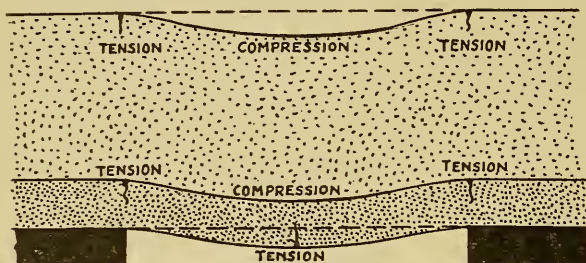


FIG. 11. TENSION AS A CAUSE OF SURFACE CRACKS.

amount of filling; and by the ratio between the depth and the lateral extent of the mine workings; as well as by geological conditions in general.



FIG. 12. SURFACE CRACKS IN WESTERN PENNSYLVANIA.

2. *Surface Cracks.*—The surface cracks and fissures that appear commonly when mining is carried on at shallow depths may be due to one of several causes. As the mine roof sags over an excavated area the bending action produces compression in the upper part of the strata

near the center of the basin or sag, while around the rim of the basin the upper strata affected are in tension which may be sufficient to cause the surface to break or crack. (Fig. 11.) If the movement is an extensive one and if the height of the surface above the axis of bending is great, the width of the fissure may be considerable. Fissures 2 feet wide have been noted in Illinois and in West Virginia. Fig. 12 shows such surface cracks in western Pennsylvania.

The formation of surface cracks by tension is well demonstrated by an occurrence in Ashland in the anthracite district of Pennsylvania.

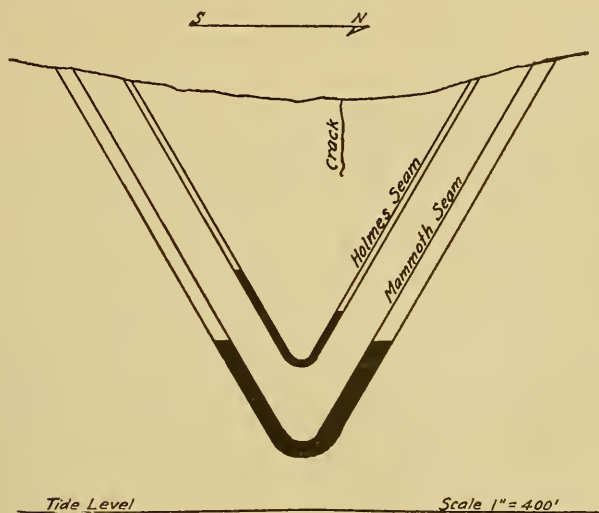


FIG. 13. SURFACE CRACKS AT ASHLAND, PA.

The crack (Fig. 13) extended for a distance of about a quarter of a mile, and was from an inch to six inches wide, causing considerable damage to property. The vertical distance to the first coal seam was over 800 feet, and later development showed that the crack did not extend to the coal. The coal along the outcrops on both the Holmes and the Mammoth seams had been removed and it is presumed that the crack was due to tension resulting from the settling of the overlying beds into the worked-out portion.*

The importance of the effect of surface beds upon draw or pull has been pointed out by A. Sopworth.† According to his observations the following classification of overlying beds may well be made:

*Foster, R. J. Discussion of Paper, Proc. Coal Mining Inst. of America, p. 147, 1912.

†Proc. Inst. C. E., Vol. 135, p. 165, 1898.

(1). "Measures consisting of fairly equal proportions of rocky and argillaceous beds, and containing thick beds of sandstone.

(2). "Measures including a small proportion of rocky beds, say 15 per cent, and only thin beds of sandstone.

(3). "Variations between these two."

In the first case the edge of the subsidence will follow or lie over the excavation and in the second case it will lie over the solid coal. In the third case the draw will vary between (1) and (2).

Kay* has emphasized the serious effects which may result from the "pull over" or draw. In his opinion this may cause much greater damage than the actual downward movement. "The strata appear to bend over the goaf in a curve of radius depending on the depth, and thereby subject the strata overlying the recently-worked area to a strain (rendered passive from the movement of the face), coincident with the progress of the working face, and, owing to its great radius and slow movement, doing very little damage to surface structures of ordinary character, as a rule." If the advance of the face is stopped, buildings over the line of the face may be seriously damaged.

R. E. Cooper† called attention to the absence of pull where the overlying beds include strong layers of limestone, shale and sandstone.

Surface cracks may be due to shear. Cracks caused in this way generally are parallel, but they may constitute more than one system. If there are two systems of fissures, generally the openings due to one system are larger and more regular than those due to the other system.

Cracks may be caused by sliding of surficial material particularly where the topography is rough. The shifting of beds of clay may cause subsidence and form a sag or basin, around the perimeter of which tension cracks will appear.

3. *Pit-holes or Caves*.—When the mining is carried on at a shallow depth where there is very little solid rock cover, or when the roof fails under shear, the movement frequently causes a sharp break in the surface, forming pit-holes or caves. (Fig. 14.) Such holes may be caused by the surficial material running into the mine entries beyond the point at which the break actually occurred.‡ This type of disturbance is the cause of much damage to the anthracite mines of Penn-

*Kay, S. R. "Effects of Subsidence Due to Coal Workings," Proc. I. C. E., Vol. 135, p. 117, 1898.

†Proc. I. C. E., Vol. 135, p. 133, 1898.

‡Fig. 14 is from a photograph of pit-holes in Indiana. In this case, coal 6 to 7 feet thick had been mined at a depth of about 100 feet. The overburden consisted of about 10 feet of shale and 90 feet of drift.

sylvania.* In regions where the surface is valuable for agriculture and for building sites, pit holes are frequently a serious problem because the cost of filling may be great. Subsidence of filled material is likely to continue for some time and the value of such filled ground for building sites is generally low.

Effect of Unwatering Surficial Beds.—Considerable discussion has been aroused by the suggestion that the unwatering of water-bearing beds of clay, marl, and sand may result in subsidence, when no mining



FIG. 14. CAVE IN SOFT SOIL. (Photo by H. I. Smith, U. S. Bureau of Mines.)

has been done. German engineers have had to contend with heavy beds of marl overlying the coal, and have made a number of observations upon the effect of unwatering the surficial beds. There is a difference of opinion, but possibly the majority of the German engineers have thought that unwatering will cause subsidence. It was held by many that when the surficial beds are drained by boreholes or excavation there is a reduction in volume of the beds and that sinking of the

*Bunting, D. "Limits of Mining Under Heavy Wash," Amer. Inst. Min. Eng., Bul. No. 97, p. 1, 1915.

surface will result. The mining industry was held responsible for surface damage, simply because it was acknowledged that unwatering had taken place.

In studying the subsidences about Essen in 1866 and 1868, von Dechen came to the conclusion that the subsidences and surface cracks were not directly the result of the coal workings, but that they were caused by the partial drying of the chalk marl and green sand overlying the coal measures which was caused by unwatering through the mines, boreholes and wells. He also pointed out that there was a shrinkage in volume in the chalk marl, due to the dissolving of carbonate of lime in the marl.

Later investigations led the German engineers to change their views upon the effect of unwatering. Graff made tests and showed* that drainage does not cause any changes in volume in gravel, sand, and quicksand. He concluded that subsidence will not result from unwatering if no solid material is carried away mechanically.

Tests made in the laboratories of the United States Bureau of Mines at Pittsburgh have shown that materials flushed with water do not compress nearly so much as the same material if dry. This would seem to indicate that by unwatering the strata of a mineral deposit, damage may be caused to the surface, even though no solid material is carried away.

F. Bernardi holds that the drying of beds of sand does not cause a decrease in volume or a reduced bearing power.† He reached this conclusion because in "water-soaked sand strata, the grains of sand rest upon grains of sand, and the weight of the surface is carried by these grains of sand resting upon one another and not by the water." If the drying of sand causes a decrease in volume, the wetting of sand should cause an increase.

Of the Austrian engineers Rziha held that unwatering may cause subsidence but the later writers, as Jicinsky and Goldreich,‡ who have had a better opportunity to make observations, hold that no movement occurs if the water does not carry away any solids mechanically or in solution. Data on the shrinkage of beds of loam and clay have been assembled by R. Dawson Hall.§ "A clay slime, 200 feet thick, will reduce to 50 feet and less, as a result of drainage, and though such a

*Graff "Verursacht der Bergbau Bodensenkungen durch die Entwässerung Wasserführender diluvialer Gebirgs schichten," Glückauf, 1901.

†Kolbe, E. "Translocation der Deckgebirge durch Kohlenabbau," p. 63.

‡Goldreich, A. H. "Die Theorie der Bodensenkungen," p. 15, Berlin, 1913.

§Hall, R. Dawson. "Data on Petro Dynamics," Mines and Minerals, Vol. 31, p. 505, 1910.

result is rare, . . . , yet the figures suggest what an action drainage has in shrinkage of roof coverings of mines and how even clays of great age may lose bulk by mining operations and let down the rock or surface with its buildings above them. German and English investigations* have been made of the shrinkage in air of flint clays. A flint clay drying in air will shrink in all directions 5 per cent, so that it will measure linearly only 95 per cent as much as before shrinkage. The loss in drying is 14.26 per cent, and this, if the clay were plastic, so as to give laterally with freedom, would reduce the thickness of the bed 14 feet 3 inches in every 100 feet of depth of measure."

4. *Effect on Drainage.*—In the prairie lands and the river-bottom lands of the coal fields of the Middle West, the complete removal of the coal from horizontal beds at comparatively shallow depths has been attended with the problem of the drainage of the surface. Over large areas of prairie land there may be almost no natural drainage, and if the mining of several feet of coal permits the uniform subsidence of the surface, large sheets of water may stand for a number of months over the subsided land, thereby greatly reducing its value for farming purposes. In many instances (as will be noted fully later) the value of the land for farming purposes exceeds greatly the value of the coal in the ground at the present leasing rates.

Satisfactory artificial drainage has been provided in such flat prairie land by the laying of drain-tile at considerable expense. Subsidence may seriously disturb this tiling and may make the entire drainage system of little or no value. In a district such as the Mississippi Valley, where the streams are bordered by extensive bottom lands that are little if any above the high water line, it is claimed that surface subsidence may materially increase the area flooded at a time of high water and may even produce areas that are continually under water or are too wet for farming purposes.

5. *Effect on Water Supply.*—Subsidence of strata generally results in the formation of cracks and fissures in the rock which may be sufficient to permit the escape of water from a water-bearing bed which may have been the source of the water supply of a community or of an industry; thus the fissuring of the rock beneath gravel beds may permit the drainage of the beds which have been the source of water.

Numerous wells and cisterns have been damaged permanently by subsidence due to mining. Instances of only a temporary loss of water

*Reis, H. U. S. Geol. Survey, 19th An. Report, pp. 404-406, 1897-98.

in wells have been noted in Illinois, Oklahoma, Maryland, and Pennsylvania, the wells furnishing the normal supply of water after subsidence has ceased if below the wells there are beds of such texture that the fissures will close tightly enough to hold water.

SUBAQUEOUS MINING.

The subaqueous mining of coal and other minerals may shatter the overlying strata and permit an inrush of water which will destroy life and property. A number of valuable mineral deposits have been opened at the edge of the ocean, and from time to time the workings first made on the shore portion have been extended seaward until the mining of the under-sea portion by a safe method has become the chief problem in the undertaking.

Much attention has been given to the study of pillars and of subsidence owing to the vital necessity of mining in such a manner that water may not enter the mine. Particulars regarding the working of coal seams under the water of oceans, rivers and lakes are given in table 2:*

England, Scotland, and Wales.

Coal is being worked under the sea along the coasts of the counties of Northumberland, Durham, Carmarthenshire and Flintshire in England and Wales and also to some extent off the coast of Linlithgowshire in Scotland.† The coal beds dipping under the Firth of Forth have been mined extensively. Here there are a number of faults parallel to the shore which drop the seams on the seaward side. The bed of the Firth of Forth, although very deep at places, is covered first by a stratum of very hard, stiff unstratified till or boulder clay, which covers the solid rock, while above this is a deposit of reddish plastic clay, from 30 to 40 feet thick and in places finely laminated. This covering forms a waterproof barrier and prevents the sea from reaching the underlying strata. There are four important coal seams having a total thickness of about 15 feet. The lowest one lies at a depth of 340 feet at the shaft and dips rapidly seaward. "Operations of late years have shown that seams can be worked on the longwall system under the sea, with faces from 4 to 8 feet in height, at depths which are small in comparison with those of the workings in most modern collieries. The seams have been worked in three instances to their outcrop against

*Atkinson, A. A. Trans. Inst. Min. Engrs., Vol. 23, Appendix IV, p. 644, 1901.

†Atkinson, A. A. Trans. Inst. Min. Engrs., Vol. 23, p. 622, 1901.

TABLE 2.

PARTICULARS OF COAL SEAMS WORKED UNDER THE WATERS OF OCEANS,
RIVERS, AND LAKES.

No.	Name of Colliery	Name of Coal Seam Being Worked	Depth Below the Water Feet	Thickness of Coal Seam Feet	Water of Oceans, Rivers, and Lakes
1	<i>New South Wales</i> Australian Agricultural Company, Sea Pit or New Winning.....	Borehole	145-190	15 to 16	Pacific Ocean (a)
2	Hetton	Borehole	300	7 to 8	Pacific Ocean (b)
3	Hetton	Borehole	165-300	7 to 15	River Hunter (c)
4	Newcastle Coal Mining Company, A and B Pits	Borehole	145-170	6 to 11	Pacific Ocean (d)
5	Stockton	Borehole	240-260	7 to 8	Pacific Ocean (c)
6	Wickham and Bullock Island	Borehole	120-260	6 to 16	River Hunter and Throsby Creek (f)
7	<i>Cumberland</i> Harrington	Main Band	126-168	6½	Irish Sea (g)
8	Workington*		90	10	Irish Sea (h)
9	<i>Northumberland</i> North Seaton and Cambois	Low Main	360	4 to 6	German Ocean (i)
10	<i>Durham</i> Ryhope**	Maudlin	1,830	7	German Ocean (j)
11	Seaham**	Maudlin	1,830	5	German Ocean (k)
		Hutton		4½	German Ocean (l)
12	Wearmouth**	Maudlin		5½	German Ocean (m)
		Hutton		4½	
13	Whitburn**	Maudlin			(n)

*Dunn, M. "A Treatise on the Winning and Working of Collieries," p. 230, 1848.

**Information derived from Mr. Thomas Bell's "Notes on the Working of Coal Mines Under the Sea, and Also Under the Permian Feeder of Water in the County of Durham." Transactions of the Manchester Geological Society, Vol. 36, pp. 366 to 399 and 554 to 559, 1899.

(a) Workings extend about 600 feet beyond high-water mark. The pillars are 96 feet by 36 feet, the bords are 18 feet wide and the cut-through or cross-holings 9 feet wide. About 4 feet of top coal is left next to the roof.

(b) Workings extend about 500 feet beyond high-water mark.

(c) The pillars are made 90 feet by 24 feet; the bords are 18 feet wide, and the cut-through or cross-holings 9 feet wide.

(d) The workings extend about 800 feet beyond high-water mark, including the winings. About 3 feet of top coal is left next to the roof and a little bottom coal is also left.

(e) Workings extend about 2,500 feet beyond high-water mark. The pillars and bords are of the same dimensions as those at the Hetton colliery.

(f) The pillars and bords are of the same dimensions as those at the Hetton colliery.

(g) About 2 feet top coal and slate is left on, next to roof, in narrow places. The minimum thickness of cover has been fixed at 126 feet. The pillars are left 57 feet by 52 feet, the bords are 14 feet wide, and the walls or cross-holings are 9 feet wide. Thus, 32 per cent of coal is worked, and the pillars are not crushed. About 66 per cent of the overlying strata is compact sandstone. Feeders of water, occurring in workings where minimum cover had been reached, have since become quite dry.

(h) The workings extend 4,500 feet under the sea. The bords were 15 feet wide and the pillars 21 to 24 feet thick. The manager, in order to increase the output of coal, commenced to rob the pillars, this resulting in falls and feeders of salt water. Warnings were given as to what would happen, but these were unheeded. On July 30, 1837, the sea broke in and 36 men and boys and a number of horses were drowned, and the colliery irrecoverably destroyed.

(i) At Cambois bord-and-pillar longwall is being worked under the sea and headings are driven 300 feet in advance to ascertain the existence of any fault or break in the strata.

(j) The workings extend 5,500 feet from low-water mark under the ocean, and over 400 acres of goaf have been formed.

(k) The workings in the Maudlin seam extend 5,000 feet under the ocean and about 85 acres of goaf have been formed.

(l) Twelve pillars, each 120 feet by 90 feet, have been removed in this seam under the goaf of the Maudlin seam, rising seawards from 2 to 2½ inches per yard for the last 1,200 feet.

(m) Workings extend 4,000 feet under the ocean.

(n) Workings extend 4,000 feet under the ocean.

the boulder clay, at depths from 137 to 400 feet below high-water mark, without any accident."* The thickness of the cover under which the whole of the coal seam has been mined is less in this mine than in any other submarine mine in Great Britain.

The workings extending farthest seaward are reported to be those at Whitehaven, which at the William pit extend under the Irish Sea a distance of 19,000 feet (1901) from high-water mark. The coal seam is 10 feet thick and is worked by rooms 18 feet wide with pillars 75 feet square. There is also a higher seam about 7 feet thick which has been worked in places.† North of the William pit is an old mine which has been flooded.

The mining of under-sea coal will become a very important matter in time in Scotland.‡

Restrictions have been imposed upon the working of Crown coal in Great Britain. In the case of one colliery the working of coal under the ocean, unless there is at least 126 feet of strata between the bed of the sea and the top of the seam, and the removal of pillars or the adoption of the longwall system, where there is less than 360 feet of intervening strata are prohibited. Under specified conditions the entire removal of the coal-seam is permitted where the minimum thickness of cover is 270 feet.§

It has been advised that the workings of coal on the Northumberland coast be limited to areas where there is a minimum of 270 feet of solid strata above the seam. The bed of the ocean generally consists in this vicinity of a stiff clay.¶

Australia.

In New South Wales coal mining has been carried on extensively beneath the River Hunter, the Pacific Ocean, and its tidal waters.** Four seams have been worked in parts of this area, the total thickness ranging from 19 to 43 feet. Operations in the vicinity of the outcrop are dangerous because channels in the coal measures become eroded by old streams, and later these channels become filled with alluvial deposits. In general, the coal measures dip slightly toward the ocean,

*Cadell, H. M. "Submarine Coal Mining at Bridgeness, N. B." Trans. Inst. Min. Engrs., Vol. 14, p. 237, 1897.

†Moore, R. W. Trans. Inst. Min. Engrs., Vol. 23, p. 660, 1901.

‡Atkinson, J. B. Trans. Inst. Min. Engrs., Vol. 14, p. 253, 1897.

§Atkinson, A. A. Trans. Inst. Min. Engrs., Vol. 23, p. 629, 1901.

¶Robertson, J. R. M. Discussion. Trans. Inst. Min. Engrs., Vol. 28, p. 133, 1904.

**Atkinson, A. A. "Working Coal Under the River Hunter, the Pacific Ocean, and Its Tidal Waters, Near Newcastle, in the State of New South Wales." Trans. Inst. Min. Engrs., Vol. 23, p. 622, 1901.

but there are many local dips and faults. The usual dip is given as 1 in 36. There are thick deposits of clay covering the outcrops in places.

Owing to the weakness of the roof a number of inundations have resulted at inshore mines from letting down the sand overburden. In consequence of a fall of roof there was a rush of water into the Fern-dale Colliery in 1886 and a miner lost his life.* A commission was appointed to investigate this accident and the report submitted included a review of conditions at all the collieries in the district. The title to the coal beneath the River Hunter and the tidal waters resides in the Crown and the leases to these coal lands now include regulations controlling the method of mining beneath bodies of water, with the view of protecting life and also of preventing large volumes of water entering old workings, and thereby interfering with the mining of the coal in the adjacent area.

The mines of the district use the pillar-and-room system. The dimensions of pillars and rooms vary, but in general 50 per cent of the coal is recovered. The practice in a number of the mines is to drive 18-foot rooms, leave 24-foot pillars, and recover part of the pillar coal. When the pillars were left only 18 feet wide on first mining a number of crushes resulted. Owing to the presence of thick, impervious beds of clay no water entered the mines where these crushes occurred, although at equal depths on land the crushes caused surface subsidence and some damage to buildings. In one of the mines the rooms are 18 feet and the pillars 36 feet.

The quantity of water being pumped from the mines varies from 50 to 600 gallons per minute and in most places this water is decidedly salty. Vertical boreholes are put up to determine the thickness and character of the overlying beds.

In determining the safe working limit under the ocean the following conditions have been considered:

- (1) The character of the overlying strata, with special reference to loose deposits of alluvium or beds of clay between the bed of the ocean and the coal seam.

- (2) The presence of faults and dykes in the strata.

- (3) The dimensions of pillars to be left and the width of openings to be made.

- (4) The utility of leaving coal next to the roof in some cases.

*New South Wales Royal Commission on Collieries. Report on the Accidents at Fern-dale Colliery, p. 17, Sydney, 1886.

The special conditions of working under tidal waters prescribed in the leases are notably as follows:

(1) The maximum width of rooms shall be 18 feet and the minimum width of pillars 18 feet.

(2) The pillars 18 feet wide shall not be removed.

(3) All headings and rooms shall be driven on sights.

(4) All workings shall be surveyed accurately every three months. All dates of working must be shown on the plan.

(5) The plan of the mine shall contain a faithful record of all dykes, fissures, etc., and shall indicate all excavations as they actually exist.

(6) In one road of every pair of leading headings, a borehole shall be kept going 10 feet in advance, and all leading headings shall be driven at least 150 feet in advance of the working rooms.

(7) When dykes or fissures are stuck in the boreholes, precautions must be taken to protect against possible danger which may result from weakness of roof or flow of water when the dykes or fissures are penetrated by the heading.

(8) The coal under the ocean should not be attacked until after a large goaf has been made by extensive workings under the mainland.

(9) The most accurate information available shall be obtained as to thickness and character of the strata and estuarine deposits overlying the coal seam before commencing to work it.

Similar conditions are specified for working under the sea except as follows:

(1) The minimum width of pillar shall be 24 feet.

(6) All leading headings shall be driven at least 300 feet in advance of the working rooms.

(9) Boreholes penetrating the roof for a height of 30 feet above the coal seam shall be driven on the leading headings 300 feet in advance of the work and 60 feet apart.

Newfoundland.

At Wabana there is a series of iron ore beds which lie in a synclinal trough, one edge of which passes through Belle Isle. The three uppermost beds are mined in both the land and in the submarine areas. The ore beds pass beneath Conception Bay and apparently outcrop in the floor of the bay. The center of the basin is estimated to be about three miles from shore. The lowest bed is from 15 to 30 feet thick.

The method of mining is pillar and room, the rooms being 250 feet long and turned on 35-foot centers with 20-foot pillars.

The development in the submarine territory is sufficient to allow an annual output of 1,000,000 tons, and the total ore reserve has been estimated at practically 400,000,000 tons after proper allowance was made for pillars, faults, and poor zones. The principal ore bed outcrops on Belle Isle and dips seaward so that at high water mark it has a depth of 70 feet; at 3,000 feet from shore the bed is 268 feet deep and has 180 feet of cover. The average grade of the slope is 16 per cent.* According to a private communication from E. E. Ellis, Geologist, Tennessee Coal, Iron & R. R. Co., Birmingham, Ala., 1913, the longest slope at the Wabana mines was 7,500 feet and the end was 6,000 feet under the water.

Cape Breton Island.

The coal measures of Cape Breton Island extend under the ocean, and a number of the coal seams have been worked in these submarine areas. The measures dip at a steep angle, while the sea floor dips at a moderate angle so that the thickness of cover increases rapidly. Owing to the rapid erosion of the outcrop by the sea, some of the seams have been lost. The Mabou mine was flooded from the ocean,† because of a break in the roof in 1909, and the Port Hood Colliery was lost by a flood resulting from the entrance of water through a feeder which was opened when pillars were extracted at a point where 942 feet of solid strata were supposed to lie between the coal seam and the floor of the ocean.

The workings of some of the companies have already been extended seaward a distance of $2\frac{1}{2}$ miles from high-water mark and it is probable that in the future a large part of the coal output will be obtained from these submarine fields. The government has prescribed regulations to control the size of openings and methods of working under shallow cover.‡ Where the cover is less than 180 feet the coal may not be mined; mine openings may be driven where there is not less than 100 feet of cover. Where there is less than 500 feet of solid cover the workings must be divided into sections not more than one-half mile square and a coal barrier not less than 90 feet thick must be left around each section. The barrier may be pierced by not more than four openings, not more than 9 feet wide by 6 feet high. In 1904

*Cantley, F. "Wabana Iron Mines." Canadian Min. Inst., Vol. 14, p. 274, 1911.

†"Coal Mines Under the Sea," Coll. Eng., Vol. 34, p. 17, 1913.

‡Coal Mines' Regulation Act of 1912, Sec. 54, Nova Scotia.

the government mine inspectors and the management of the Dominion Coal Company agreed upon the size of pillars to be left in the mining of submarine coal.*

The dimensions of rooms and pillars, the percentage of coal left in the form of pillars, and the thickness of cover are shown in Table 3.

TABLE 3.

DIMENSIONS OF ROOMS AND PILLARS, DOMINION COAL COMPANY.

Depth of cover (feet)	Harbour Seam*		Hub and Phalen Seams**		Percentage of coal left in pillars
	Room width (feet)	Size of pillar (feet)	Room width (feet)	Size of pillar (feet)	
200	20	27 x 75	20	30 x 75	51
250	20	27 x 75	20	30 x 75	51
300	20	30 x 75	20	34 x 75	54
350	20	33 x 75	20	36 x 75	56
400	20	36 x 75	20	42 x 75	58
450	20	39 x 75	20	46 x 75	60
500	20	42 x 75	20	50 x 75	61
550	20	45 x 75	20	54 x 75	63
600	20	48 x 75	20	58 x 75	64
650	20	51 x 75	20	62 x 75	65
700	20	54 x 75	20	66 x 75	66
750	20	57 x 75	20	70 x 75	67
800	20	60 x 75	20	74 x 75	67
850	20	63 x 75	20	78 x 75	68
900	20	66 x 75	20	82 x 75	69
950	20	69 x 75	20	86 x 75	69
1,000	20	72 x 75	20	90 x 75	70

*Thickness mined from Harbour Seam, 6 feet.

**Thickness mined from Hub and Phalen Seams, 9 feet.

British Columbia.

A disaster† which may be compared with those occurring in subaqueous mining resulted when the workings of the old Southfield Colliery near Nanaimo, British Columbia, tapped the drowned workings of the South Wellington Mine No. 1 of the Pacific Coast Coal Company on February 9, 1915. The inrush of water resulted in the death of 20 men. It was believed that the new workings were 450 feet away from the water and it was planned that a 100-foot pillar should be left between the water and the new workings. At the time of writing this report no evidence is available showing whether or not the Coal Mines

*Dick, W. J. "Conservation of Coal in Canada," p. 35, Toronto, 1914.

†"Twenty Men Drowned in Mine Near Nanaimo, B. C." Coal Age, Vol. 7, p. 374., 1915. Watson, R. L. "Coal Mining on Vancouver Island." Mines and Minerals, Vol. 21, p. 249, 1901.

Regulation Act was being complied with; namely, that drill holes shall be kept in advance of the workings.*

Japan.

A large proportion of coal is mined under the ocean in Japan.†

The most serious accident in the whole history of subaqueous mining occurred in this country on April 12, 1915, when 237 men were killed by the flooding of Higashimisome Colliery. The mine is situated in Ube, Yamaguchi-ken and the chief production is from two beds lying wholly under the sea. The output is about 500 tons per day. Four shafts were sunk on the shore, each 119 feet deep, from which two beds are worked to a distance of about 4,000 feet from the coast.

The cause of the accident was the entering of water into the underground workings through a fault in a bed of sandstone 155.4 feet thick, above which there is an alluvial deposit of clay and sand 82.6 feet thick.

A small flow of water occurred when the fault was first reached. The final inrush followed the breaking of a hole about four feet square in the floor of an entry of the upper bed, a few feet back from the fault. Through this the water entered so rapidly that the mine was completely flooded in two hours. The quantity entering was estimated at 392,000 cubic yards. The sea bottom was lowered 60 feet over a small area showing that a considerable amount of solid matter was washed in.

The opening was apparently sealed by the solid material and it was planned that the mine should be reopened by filling the depression in the sea bottom with clay and sand, pumping out the water, and building dams to protect the workings from any future break.‡

INDUSTRIES AND INTERESTS AFFECTED BY SUBSIDENCE.

Surface subsidence involves more than the question of the present value of the land; in many instances the fundamental problem involves the relative present and future importance of various industries and interests. Among the most important of these are agriculture, transportation, and the various interests of municipalities.

*"Where a place is likely to contain a dangerous accumulation of water, the working approaching such place shall not exceed eight feet in width, or such greater width as may be permitted by the Chief Inspector of Mines, and there shall be constantly kept at a sufficient distance, not being less than five yards in advance, at least one borehole near the center of the working, and sufficient blank boreholes on each side. (British Columbia Laws, 1911, Chap. 160, Part XI, Rule 14.)"

†Trans. Inst. Min. Engrs., Vol. 28, p. 133, 1904.

‡Colliery Engineer, Vol. 36, p. 19, 1915.

1. *Agriculture*.—In the consideration of the agricultural interests involved, attention must be directed to the probabilities of subsequent use for agricultural purposes of land not tilled at present. Probably in no state where mining is important is the value of farm lands in the mining districts higher than in Illinois. It will be shown in another bulletin how the present value of these lands for mining purposes (removing all the merchantable coal) and the present value for farming purposes compare. It has been predicted that the value of the fertile lands of the "corn belt" will increase greatly in fifty years.

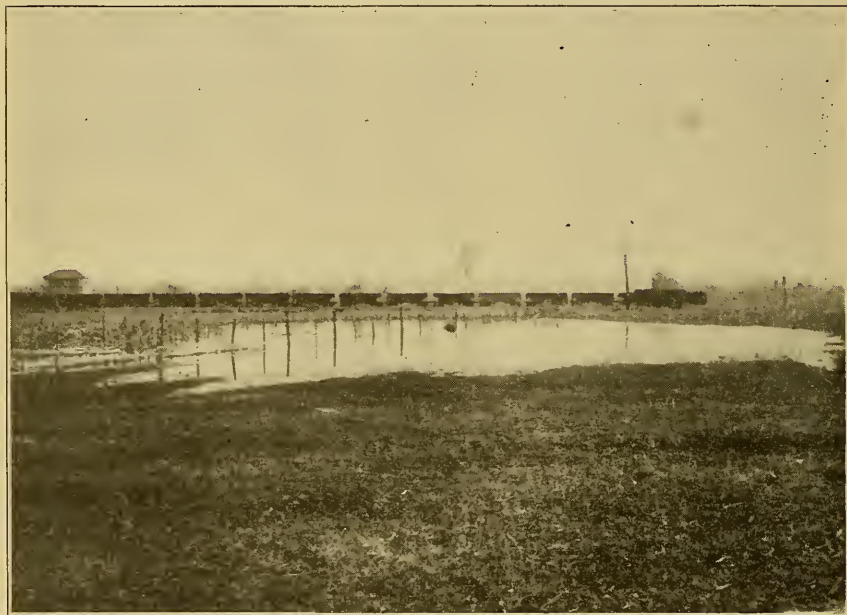


FIG. 15. POND FORMED BY SUBSIDENCE.

An agricultural expert has expressed the belief that northern Illinois land will sell for from \$400 to \$500 per acre, and the best land in the southern counties for \$200 by the year 1965.

In the longwall field of northern Illinois, where it is claimed that mining has lowered the surface so that drainage is deranged, it is estimated that large drainage projects have cost from \$15 to \$40 per acre. Fig. 15 illustrates the formation of a pond in a nearly level country by subsidence after mining. Coal of no greater thickness has been mined and is being mined in adjacent states. Estimates of the

coal resources of Illinois show that only twenty per cent of the coal occurs in beds more than four feet thick and of the total area (37,486 square miles) underlain by workable coal beds, 32,979 square miles do not contain coal more than four feet thick. Over this great area it is possible that sometime mining by the longwall system may produce subsidence unless a filling system is used that is more effective than any at present in use. This statement regarding the thin coal beds in Illinois applies as well to large areas in Michigan, Ohio, Indiana, Kentucky, Missouri, Iowa, Kansas and several other states, and it is evident that the importance of the subject of subsidence will be even greater in the future than at present.

2. *Transportation.*—Surface subsidence may interfere seriously with transportation by injury to the beds of canals and railroads and the caving of highways and streets. As previously noted, mining in Great Britain and on the continent has necessitated the raising of the banks and the filling of the bottom of many canals. In some instances, canals have been maintained on grade, while the land which they traverse has subsided as much as 20 feet. The necessity for protection of these interests has become so great that laws have been enacted which require that thirty days' notice be given of mining under railways, reservoirs, buildings, or pipes or within a prescribed distance.*

The practice regarding the protection of the right of way of railroads has differed from time to time and has varied also in different countries. The general policy in Europe seems to be to remove all the coal if possible, and the tendency on the continent is to use filling under railways in order to reduce the amount of subsidence.

In the United States many of the great railway systems do not grant the right to mine coal beneath the right of way, if the company has ever owned the coal right. However, coal has been mined under many branch lines and under some of the main lines of railroads traversing the coal districts. Fig. 16 shows the effect of one subsidence in southern Illinois. In the anthracite fields of Pennsylvania many instances might be cited of subsidence of railway tracks. No serious accidents have resulted, as the railway companies have guarded carefully all points where movement is feared. There are no laws regulating mining under railways in the United States. When a pit hole or cave extends to the surface near or under a railway track, the problem of restoration is principally a problem of filling. Good

*Cockburn, J. H. "Minerals Under Railways and Statutory Works." Trans. Inst. of Min. Engrs., Vol. 39, p. 104, 1909-10.

illustrations are found in some of the iron mines of the Lake Superior district, where extensive filling has sometimes been necessary to preserve the grade of tracks, amounting in one case to more than 50 feet.

When the movement is gradual and principally a horizontal one due to tension or compression the problem is much different. In Ger-



FIG. 16. DISTURBANCE OF GRADE BY SUBSIDENCE.

many many observations have been made upon railway track subject to tension or compression on account of subsidence over mines. In one instance, because of the crowding of the ground toward the center of the subsiding area, track 150 feet (50 meters) in length had to be shortened from 1 to 2 inches (3 to 5 cm.). Rails were buckled up or to the side, and the crowding forward of the rails and ties caused the earth or ballast to be pushed forward or crowded up and an open space appeared along one side of the tie. These spaces have been noted as much as one-third of an inch wide. In one sag in which the maximum subsidence was about 3 feet (1 meter) in five years it was necessary to shorten the rails 2.66 meters (70 cm.) in a total distance of 658 feet (200 m.). When the track was in tension the rails were stretched and at times the ends were broken.* When the principal horizontal movement is across the right of way, the trouble is easily seen on account of the effect on alinement.

The effect of surface subsidence upon bridges has been noted by

*Nolden "Influence of Mining Upon Buildings and Railways." *Elektrische Kraftbetriebe und Bahnen*, Oct. 4, 1913.

European engineers, including many British engineers.* English engineers suggest steel construction, well-tied abutments and wings, and plenty of height so that there will be sufficient clearance after the bridge has been lowered by the removal of the coal. There has been a difference of opinion in regard to the adaptability of arched or girder bridges. In the reference noted, an example is given of the mining of a seam, 7 feet 6 inches thick, at a depth of 216 feet beneath an arch of 20 feet on the main line of a railroad. The arch was not damaged by subsidence. It was conceded that arches from 50 to 60 feet long would not be advisable under similar conditions.

In 1868 several bridges were built in England on land that was known to be subsiding on account of the mining of the coal, and special precautions were taken to preserve these bridges. The rails were carried on wrought iron girders and cross girders. The foundation was carried deep enough to permit the construction of a concrete base 4 feet thick. On this base was laid two courses of elm planking, each 4 inches thick, on which four courses of brick footing were built, and on these four courses was laid a hoop iron interlaced frame, $4\frac{1}{2}$ -inch mesh, extending over the whole of the abutments and wing walls. This arrangement was repeated every four courses. Later, in some places, the foundation sank as much as 4 feet, but the whole bridge was lowered unbroken, and it was necessary only to lift the girders and the track to grade.†

Experience has shown that the damage to a bridge will be least if the workings (longwall) approach it broadside. The working face will pass under the structure much more quickly with that plan of working and there will be probably less difference in elevation between the ends of the structure at any stage of the subsidence.

In the construction of the Hull and Barnsley Railway across the South Yorkshire coal field, which it traversed for twelve miles, the problem of supporting bridges was of great importance. Owing to the great value of the coal beds, the plan of reserving coal pillars was given up. W. Shelford‡ advocated the separation of the bridge masonry into parts which could subside independently of each other, but should have the materials in each part bonded together. Several bridges were designed on this principle with abutments and wings separated only by a straight joint of mortar, which was concealed by a pilaster.

*Kay, S. R. "Effect of Subsidence Due to Coal Workings Upon Bridges." *Proc. Inst. of Civ. Eng.*, Vol. 135, p. 114, 1898.

†Lynder, J. H. *Proc. Inst. of Civ. Eng.*, Vol. 135, p. 161.

‡*Proc. Inst. Civ. Eng.*, Vol. 135, p. 164, 1898.

A large bridge built in 1884 after this plan subsided 3 feet in 1891. The wing walls separated from the abutments, but the abutments themselves were uninjured and subsided bodily, so that they were only 3 or 4 inches out of plumb. When subsidence had ceased the wings were repaired and the bridge was again placed in service.

The effect of subsidence upon railroad tunnels has been noted previously, particularly in the construction of the Merthyr tunnel in Wales, and the Greentree tunnel at Pittsburgh, Pa.

3. *Municipalities.*—As previously noted, many towns in Europe



FIG. 17. BREAK IN SIDEWALK DUE TO SUBSIDENCE.

and America have been damaged by subsidence caused by mining. The damages to property in municipalities may include:

(a) *Injury to Streets, Sidewalks, and Transportation Lines.*—When pit-holes or caves occur, it becomes necessary to fill until subsidence has ceased and then reconstruct the street upon the most satisfactory grade. When there is horizontal movement, due to tension or compression, rather than caves, the streets, curbing, and sidewalks may be crushed or heaved (Fig. 17), or there may be tension great enough to cause serious cracks. This trouble has become so severe in certain

German cities that in the sections where compression occurs the gutters and curbs are laid so as to have elastic and waterproof joints. When large gaps are left in construction between curbstones they are covered with strips of sheet iron about 2 inches wide. In order to prevent the overturning of curbing, due to compression occurring transversely, the flagging is made narrower than the sidewalks and a strip of material that will permit compression is laid between the flagging and the curb. Coherent paving, such as asphalt, cement, and concrete, is not used because it would be cracked or crushed.*

(b) *Injury to Buildings, Towers, and Chimneys.*—This may be due to caves, or to tension, compression, or twisting. Large high buildings suffer more than low buildings covering but little ground. Masonry and concrete structures are damaged more than those of wood.

E. Kolbe has discussed at some length the nature of the damages to buildings,† and has pointed out the various factors and conditions with which one must deal in preserving buildings upon land which has subsided as follows:

- (1) A building may sink wholly or in part into a surface break.
- (2) A building may stand upon the edge of a break and be suddenly and violently twisted or wrenched and shaken.
- (3) A building may be located in the mining area and may be subjected to the earth movement and be damaged by the jamming of the adjoining houses.
- (4) A building lying over the mined area may sink slowly in the subsidence basin without undergoing greater damage than being placed in an inclined position.
- (5) A building may suffer on account of the shock resulting from a fall of roof in the mine.

The types of cracks in brick buildings particularly around and between (Fig. 18) windows have been noted by Kolbe, as shown in the accompanying illustrations. As the illustrations show, the fracture generally follows the joints of the mortar, as these offer the least resistance. When cut stone window sills and lintels are used (Fig. 19), the fracture naturally follows upward around the stone without cracking it. In long brick or tile walls without openings, as for example walls (Fig. 20) surrounding estates, there may be three types of fractures in relation to direction:

*Nolden "Influence of Mining Upon Buildings and Street Railways." *Elektrische Kraftbetriebe und Bahnen*, Oct. 24, 1913.

†Kolbe, E. "Translocation der Deckgebrige durch Kohlenabbau." Essen, 1903.

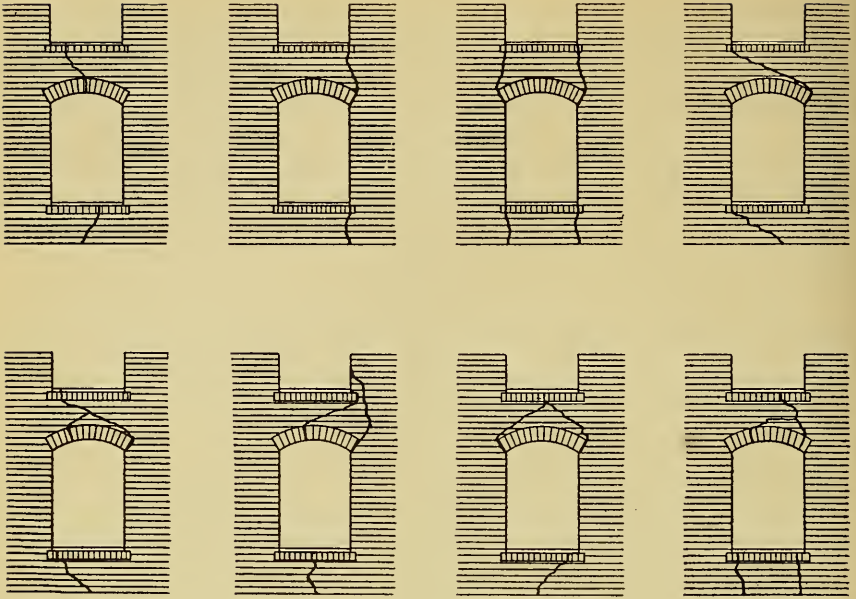


FIG. 18. CRACKS IN BRICK BUILDINGS.

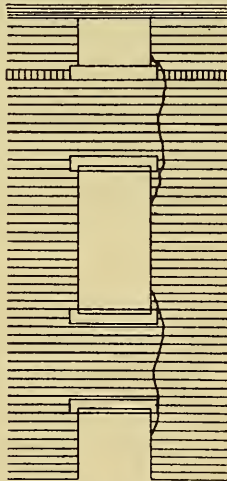


FIG. 19. EFFECT OF SUBSIDENCE ON STONE LINTELS AND SILLS.

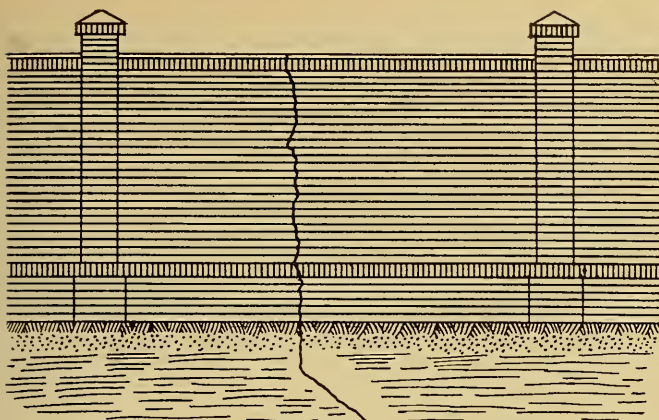


FIG. 20a.

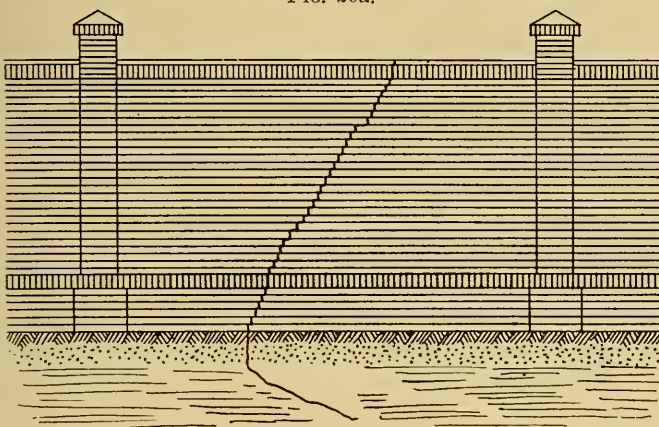


FIG. 20b.

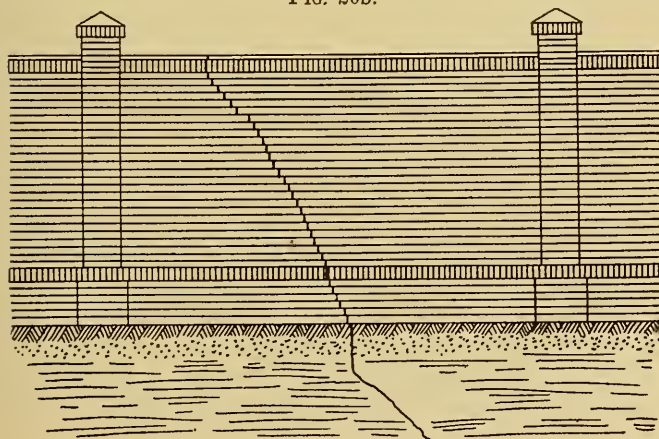


FIG. 20c. CRACKS IN LONG WALLS.

(1) The fracture may go perpendicularly up the wall and break the stone coping. (Fig. 20a.)

(2) It may extend diagonally away from the plane of the crack in the ground following the joints in the brick work. (Fig. 20b.)

(3) It may extend along the joints of the brick work diagonally in the same general direction as the plane of fracture in the ground. (Fig. 20c.) The second type is of most frequent occurrence. The same three types of fracturing are characteristic also of high enclosing walls, partition walls, and fire walls and chimneys.

Buildings may be damaged by side movement in which structures are crowded upon each other. When the mortar in masonry walls is cracked, the arches over doors and windows fail and increased pressure is thrown upon adjacent sections of the structure. When buildings are located over the edge of a pillar or on the side of a trough caused by subsidence, the cracks may extend in step fashion diagonally across a masonry wall. Secondary stresses may cause additional cracks in other directions. An example of this type of damage is shown in Fig. 21, which is an elevation of a post office. The cracks extend in the same general direction as the cracks in the ground.

In Germany, where subsidence has been anticipated, large buildings have been erected in sections from 60 to 120 feet long and these sections have been reinforced in all directions by rods and plates so that they will withstand both tension and compression. The joints between the sections have been calked with suitable material or protected with a covering. When buildings are not of great value European engineers have removed the coal as rapidly as possible and completely if possible, advancing the working face in a direction at right angles to the axis of the most important structure. When such precautions were used, the working of two 4-foot seams of coal at a depth of 600 to 780 feet in England caused practically no damage to two rows of 120 cottages.*

When the structures are important and it is estimated that the damage caused by subsidence will exceed the value of the coal, pillars may be left or filling introduced to prevent or reduce the subsidence.†

The problem of protecting important public buildings has received serious attention in Scranton, Pennsylvania. In several instances buildings have been erected on reinforced concrete piles constructed upon the

*Longden, J. A. "Effect of Coal Workings on the Surface." *Colliery Engineer*, Vol. 11, p. 5.

†Spencer, W. "The Support of Buildings." *Trans. Inst. of Min. Eng.*, Vol. 5, p. 188, 1892-93.

rock underlying shallow coal beds which had been worked by the pillar-and-room method and of which the roof had fallen or seemed likely to do so. Engineers Griffith and Conner made an inspection of the conditions of mining beneath the city and school properties. The tabulated results of their inspection indicate that some coal has been mined under

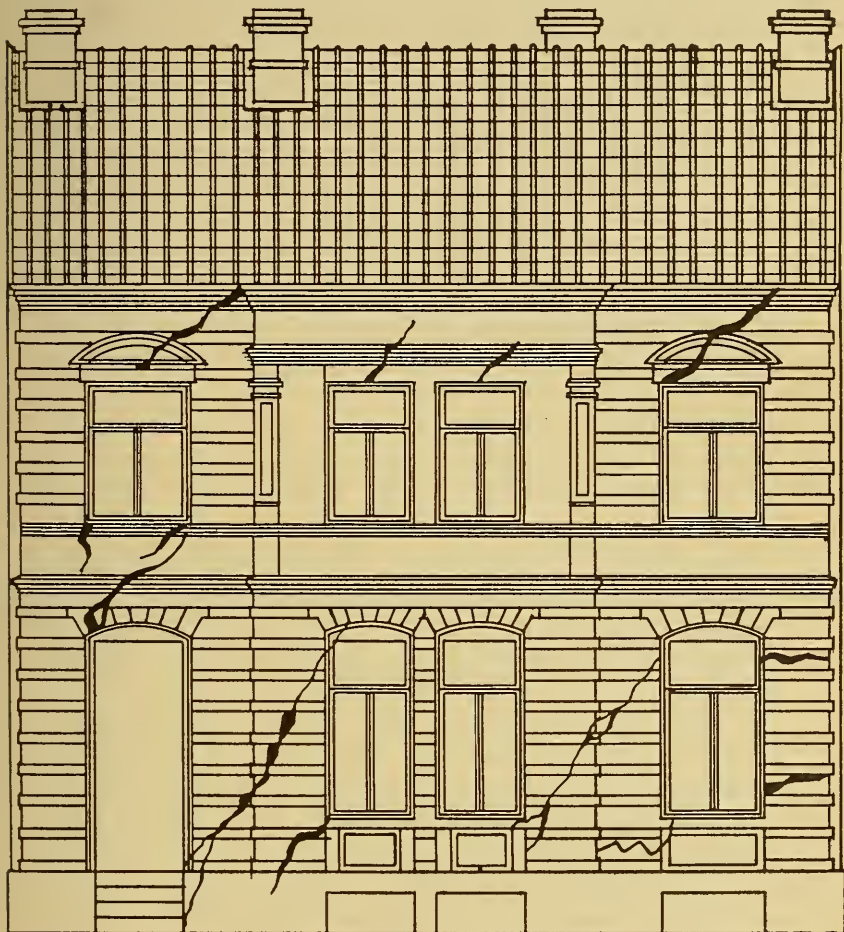


FIG. 21. CRACKS IN MASONRY WALL.

most of the buildings and that in a number of instances mining has been carried on in several beds.* Several of the buildings have been damaged by subsidence. The suggestions (*op. cit.*, p. 60) by these engineers of precautionary measures will be considered later.

*U. S. Bureau of Mines, Bul. No. 25, pp. 19-43, 1912.

The report of Enzian, Johnson, and Williams on the extent of damage done, states the results of their examination as follows:

"In the consideration of a plan which might assist in the adjustment of property damaged we considered it important to compile the following information in connection with the properties of the thirteen city blocks which have been more or less subjected to the influence of subsidences that have occurred from time to time. The total assessed valuation of these properties is \$1,430,000. The assessed valuation of the properties actually damaged is \$411,000. The estimated damage to properties actually affected is \$68,700, or about 17 per cent of their assessed valuation. The estimated damage to all the properties in the thirteen city blocks amounts to approximately 4.7 per cent of their assessed valuation. This estimate does not take into consideration any damage that may have been done to public property."*

(c) *Injury to water, gas, and steam lines.*—This type of damage is not unusual in communities in which mining has been carried on extensively. The cracking of water mains has caused damage not only through the direct injury to the main and the temporary failure of the water supply, but also through the escaping water, which in a number of instances has flooded buildings, washed out foundations, and destroyed streets, roads, and earthen structures. Fires have resulted from the escape of gas from broken gas mains. Necessity has brought about the use of expansion and compression joints of various types for preventing or reducing the damage to such lines. The need for frequent inspection of such pipe lines has made it important that they be laid in tunnels or large conduits.

(d) *Injury to sewers and sewage plants.*—Sewer lines as well as steam, water, and gas mains may suffer from subsidence, but in the case of sewer lines the difficulties are even greater, since these lines are generally constructed of materials which are less able to resist tension and compression, and a change in elevation of part of the line may render the entire system useless. An interesting experience regarding the subsidence of sewage works is reported by an English engineer, Malcolm Patterson.†

"At Ravensthorpe, in the Calder Valley, sewage works constructed in 1874 had remained intact for twenty-four years; they lay on the verge of a colliery leasehold. In August, 1897, the effluent outlet submerged

*Enzian, Johnson, and Williams "Report on Mining Conditions of the Oxford Colliery Workings, Scranton, Pa., Dec. 12, 1913."

†Proc. Inst. Civ. Engrs., Vol. 135, p. 162, 1898.

15 inches below the ordinary level of the stream into which it discharged. At his (the author's) previous visit it was at its normal level, of about 6 inches above the stream. The settling tanks were cracked across the center, and the tank sewer had settled considerably. These settlements arose from getting a 20-inch seam of coal, besides the dirt, about 150 to 160 feet deep, and the boundary of the worked coal terminated in or near the sewage workings. In the same year, a similar disturbance took place at the Castleford sewage works in the same valley. Complete re-levellings of the three roads intersecting the land were taken, and proved an average settlement of 3.3 feet throughout nine-tenths of the 12.5 acres of sewage land, without the surface being broken. In this case the getting of coal, 4 feet to $4\frac{1}{2}$ feet thick, at a depth of 603 feet, was the cause. The contour was singularly constant, the new section being almost parallel with the original section. The strata here were the shales and sandstones of the coal measures, overlaid by the marls and limestones of the Permian formation."

CHAPTER II.

GEOLOGICAL CONDITIONS AFFECTING SUBSIDENCE.

The behavior of the measures overlying the mineral deposit which is being worked depends to a large degree upon the physical character and the structure of the measures themselves. In a recent paper* before the International Geological Congress attention was called to the various geological conditions which influence the effect of underground mining upon the surface as follows:

- (1) The general character of the overlying strata.
- (2) The presence of faults, fissures, etc.
- (3) The dip of the strata.
- (4) The direction of the workings with regard to the jointing of the strata.
- (5) The compressive strength of the rocks of the various overlying beds.
- (6) The bearing power of the underlying beds.
- (7) The angles at which rocks break when stressed.

Geological conditions must be studied in each district, as no generalizations can be made which will apply without reservation to all mining fields. The measures overlying a flat seam may be made up of various beds of sedimentary rocks and in places may include sheets or beds of intrusives. The physical character as well as the thickness of each bed may vary over different parts of the same mine, and there may be faults, fissures, rolls, etc., which greatly influence the supporting power of the bed, as well as the manner in which the weight of the bed itself is distributed upon the underlying supports. Unless the thickness and the character of the beds have been proven, and unless it is known definitely that the beds are fairly uniform throughout the field under consideration, it will be impossible to formulate even approximate rules and theories regarding subsidence which will be useful in the study of the problem of surface support.

MINERAL DEPOSITS.

1. *Physical Character.*—Before considering the overlying and the underlying beds, it will be well to note some of the conditions in the deposit being worked which may greatly influence the problem of sur-

*Knox, George "Mining Subsidence." Proc. International Geological Congress, Vol. 12, p. 797, 1913.

face support. The physical character of the material being mined and of that part, if any, of the deposit which is left in the form of pillars or of filling must be considered. The texture and the structure of the rock left in pillars is of great importance in determining the burden the pillars will carry and in affecting the stability of the pillar after it has been subjected to the action of explosives in the adjacent portion of the deposit and after it has been exposed to the action of the atmosphere and water. In many coal mines, owing to the friability of the coal, it has been necessary to reduce the charges of powder used along the rib and in some instances to avoid the use of powder entirely because the pillars are more or less shattered by the force of the explosives. Rock and coal may be so weakened by jointing or cleats that the pillars offer little support. Moreover, the action of the atmosphere and moisture, particularly upon a deposit jointed as described, may greatly weaken the pillar. Soluble minerals in the pillars may be dissolved by the mine water or the moisture in the air and the pillar thus weakened. Pyrite and other minerals may be oxidized and a deterioration of the pillar will follow. It has been suggested by some that the loss of the included gas in coal beds tends to reduce the strength of the coal. The hydration or the dehydration of minerals may result in the weakening of pillars. The terms "rashing," "slacking," and "slabbing" have been applied to the process of weakening of pillars by the gradual dropping of material from the ribs, due in part to the action of moisture, oxygen, or pressure, or a combination of these agents. In mines operating in soluble minerals the preservation of pillars may be difficult owing to the flow of water in the mine or the moisture in the air. It may become necessary in mines of all types when pillars deteriorate to protect important pillars by a coating of cement or concrete.

Strength tests have been made upon coal and other minerals in order to determine how serviceable they will prove when left in pillars and in order to estimate, in advance of the opening of a mine in a new field, the minimum size of pillar which may be left in safety for the protection of the mine openings themselves and of objects on the surface.

Numerous tests have been made upon rocks used for building purposes, and the data thus secured are of service in determining the size of the pillars to be left in such rock. But more commonly the pillars left in mines are not composed of materials used for building purposes, but rather of coal, ores of the various metals, and rock mineralized more or less with substances which are not permitted in structural materials.

Moreover, the natural structural materials used are generally a selected product. Underground the pillar is frequently made up of the weakest portion of the deposit. Tests upon pillar materials are often of doubtful service, for, as a rule, they indicate the maximum load which can be borne by a unit of the mineral and one that is often a selected unit. A coal bed, for instance, is composed of layers of varying hardness, and frequently it contains streaks of mother coal that would not be included in a sample tested for crushing strength.

TABLE 4.
COMPRESSION TESTS OF ILLINOIS COAL FEBRUARY 6, 1907.
Laboratory of Applied Mechanics, University of Illinois.

Laboratory No.	Specimen from	Equivalent Section, Inches		Height, Inches	Maximum Load	
		Top	Bottom		Pounds	Lb. per Sq. In.
12401	Penwell Coal Co., Pana, Ill.	11¾x12	11¾x12	12½	316,000	2,090
12402	Empire Coal Co.	15 1/5x17 3/5	15 x15 1/3	11.3	540,000	2,170
12403	W. W. Williams, Litchfield, Ill..	13¼x13¼	14 x14	14½	186,000	1,000
12404	Herdien Coal Co., Galva, Ill.	11½x17¼	16 x13	12	208,000	1,020
12405	T. H. Watson, Litchfield, Ill..	13¾x12	13¾x12	15	224,000	1,360
12406	C. N. & V. Coal Co., Streator, Ill.	11¾x 9¼	11 x11¼	13	140,000	1,280

Tests were made in the Laboratory of Applied Mechanics of the University of Illinois upon samples of Illinois coal furnished by the Illinois Geological Survey.* The data regarding the samples and the results of the tests are given in Table 4.

	Dimensions in Inches	Area in Sq. In.	Crushing Strength per Sq. In.
Sample M—			
Parallel with cleavage.....	2.01 by 2.02	4,060	3,170
Right angles to cleavage.....	1.75 by 1.70	2,975	2,970
Sample B—			
Parallel with cleavage.....	1.95 by 2.01	3,925	3,430
Right angles to cleavage.....	1.92 by 1.93	3,802	3,050

Tests were made by the H. C. Frick Coal Company upon samples of coal from the Pittsburgh seam. These are particularly interesting as they show the strength when compression is parallel to the cleavage and also when it is at right angles to it.

*Talbot, A. N. "Compression Tests of Illinois Coal." Ill. State Geol. Sur., Bul. No. 4, p. 198, 1909.

A series of tests was made upon Pennsylvania anthracite during 1901 and 1902 by a committee from the Scranton Engineers' Club. In all 416 samples were tested. The samples were uniformly 2 inches square, but were of three different heights: namely, 1 inch, 2 inches, and 4 inches. The results are given in pounds avoirdupois per square inch of horizontal area as presented in the following summary:

Samples	Height of Sample	Grand Average as Per Sq. In.		Number of Tests
		First Crack	Maximum Load	
Northern Field	1	3022	6241	122
	2	2025	4087	116
	4	1875	2854	113
Eastern Middle Field.....	1	4996	7417	7
	2	3343	3857	6
	4	3413	3821	7
Western Middle Field.....	1	3001	8631	3
	2	788	3499	3
	4	1440	2447	3
Southern Field	1	1124	3814	12
	2	1099	2377	12
	4	988	1809	12

From the data obtained the following conclusions have been drawn: "That the squeezing strength of a mine pillar of anthracite whose width is twice its height is about 3,000 pounds to the square inch, and the crushing strength about 6,000 pounds per square inch, or, approximately twice as much. And in general, other things being equal, the crushing strength of mine pillars would vary inversely as the square root of the thickness of the bed.

"The same general rule apparently holds true also for the squeezing strength in all cases in which the height of the pillar is less than the width. In tall pillars, having a height greater than their width, the squeezing strength apparently remains nearly constant while the crushing strength continues to diminish with height according to the foregoing rule."*

Subsequently additional tests were made at Lehigh University on samples of anthracite and of bituminous coal.† Forty-five anthracite specimens were tested. "There seems to have been no uniformity in the amount of compression of the specimens taken as a whole or between the specimens from the same seam." The results of the tests upon twelve bituminous specimens were more uniform. The crushing strength

*Mines and Minerals, Vol. 23, p. 368, 1903. U. S. Bureau of Mines, Bul. No. 25, Appendix, 1912.

†Daniel, J., and Moore, L. D. "The Ultimate Crushing Strength of Coal." Eng. and Min. Jour., Vol. 84, p. 263, 1907.

per square inch ranged from 584 to 1,583 pounds, but nine ranged from 1,000 to 1,538 pounds. All of the bituminous specimens were taken from the Pittsburgh seam. Additional data on the crushing strength of anthracite coal have been secured by Bunting* and Table 5 shows the crushing strength and the relation between prism strength and cube strength.

TABLE 5.
AVERAGE RESULTS OF TESTS ON ANTHRACITE SPECIMENS.

Name of Company	Ratio $\frac{h}{b}$	Crushing Strength Lb. per Sq. In.	Prism Strength Cube Strength
P. & R. C. & I. Co.			
(25 specimens)	1	2,393	1.00
(25 specimens)	2	2,296	0.96
L. V. C. Co.			
(13 specimens)	1	1,982	1.00
(13 specimens)	2	1,591	0.80
(13 specimens)	3	1,405	0.71
L. & W-B. C. Co.			
(20 specimens)	0.71	3,025	1.22
(20 specimens)	1.07	2,566	1.00
(20 specimens)	1.24	2,393	0.87
(20 specimens)	1.43	2,008	0.81
(20 specimens)	1.77	2,090	0.76
(20 specimens)	2.06	1,880	0.84
D. & H. C. et al.			
(146 specimens)	0.50	5,113	1.63
(146 specimens)	1.00	3,131	1.00
(146 specimens)	2.00	2,234	0.71

b=Least lateral dimension.
h=Height of prism.

The crushing strength of some British coals has been measured and reported by Henry Louis.† McNair in discussing the conditions of deep mining in the Lake Superior District‡ refers to the crushing strength of the trap rock left in pillars as 1,200 tons per sq. ft. Richardson|| gives a table of the compressive strength of quartzite cubes taken from the depths of from 1,000 to 3,500 feet in Rand mines. The first fractures appeared in the specimens under a pressure of 1,945 to 6,804 pounds per square inch. The crushing strengths of these specimens were 8,054 and 9,029 pounds per square inch, respectively.

2. *Extent and Dip of Deposit.*—The problem of surface support is naturally different in the case of a deposit which underlies an area of great lateral extent from that of support when the lateral extent is small. When the deposit underlies a small area the geological structure

*Bunting, D. "Chamber Pillars in Deep Anthracite Mines." Trans. Amer. Inst. Min. Engrs., Vol. 42, p. 236, 1911.

†Trans. Inst. Min. Engrs., Vol. 28, p. 319, 1904.

‡Eng. and Min. Jour., Vol. 23, p. 322, 1907.

||Richardson, A. "Subsidence in Underground Mines." Jour. Chem. Met. and Min. Soc. of S. Africa, Mar., 1907. Eng. and Min. Jour., Vol. 84, p. 196, 1907.

of that area may be worked out fairly accurately and precautions may be taken to protect important structures on the surface.

The dip and the position of the deposit may greatly modify the necessity for and the general policy of surface support.

3. *Uniformity of Mineral Deposit.*—If there is fair uniformity in thickness, structure, quality, and depth over a large area, a systematic plan of support may be adopted, including, for example, pillars of uniform size at regular intervals or a complete removal of the deposit with or without filling. If there is not regularity as to these conditions, it becomes more difficult to employ a system of support or of working which will be economical and at the same time provide support for the surface. Notable examples of such conditions may be found in some of the coal fields of Illinois where rolls, horsebacks, and faults complicate mining, and in some of the lead and zinc fields of the Mississippi Valley, where pillars of barren rock are left in the mines and the rich portion of the deposits is mined out as completely as possible under such conditions. The pillars as a rule are neither uniform in size nor uniformly spaced. While such irregularities in the mineral deposit interfere to a degree with systematic working, yet they at times assist materially in preventing or checking extensive underground movements or subsidence.

UNDERLYING ROCKS.

The physical character of the rocks immediately underlying the mineral deposit is of great importance. Frequently coal beds are underlaid with beds of clay of such consistency that it will not support the pillars when the weight upon them is increased by the opening of rooms. The pillars are slowly pushed into the clay while the clay is forced into the rooms which have been mined. Similarly, when water reaches clay beds underlying the coal, the clay may be softened and forced into the rooms by the weight of the pillars, and a subsidence results. The term "creep" is very commonly applied to such a movement.

Very few tests have been made upon the bearing power of the clays occurring in mines, but numerous tests have been made upon clays and soils upon the surface. Owing to the importance of not placing upon the clay floor of a mine a burden which shall exceed the bearing power of clay, which is usually much less than the compressive strength of coal, the following values are of interest:*

*Baker, I. O. "A Treatise on Masonry Construction," p. 842, 10th Ed., 1912.

Kind of Material	Safe Bearing Power in Tons per Sq. Ft.	
	Minimum	Maximum
Rock—the hardest—in thick layers in native bed.....	200.0	
Rock equal to best ashlar masonry.....	25.0	30
Rock equal to best brick masonry.....	15.0	20
Rock equal to poor brick masonry.....	5.0	10
Clay in thick beds, always dry.....	6.0	8
Clay in thick beds, moderately dry.....	4.0	6
Clay in soft beds.....	1.0	2
Gravel and coarse sand, well cemented.....	8.0	10
Sand, dry, compact, and well cemented.....	4.0	6
Sand, clean, dry.....	2.0	4
Quicksand, alluvial soils, etc.....	0.5	1

The data on clay given in the table are not for fireclay, and no data have been obtainable which are the results of observations upon the supporting power of such clay of the character and occurring under conditions similar to those found in coal mines.

OVERLYING ROCKS.

The study of subsidence due to mining operations involves particularly a consideration of the rocks overlying the mineral deposit. Lack of uniformity in the overlying measures is the rule, not the exception, and this fact must be recognized in all attempts to formulate theories and rules. The effect of different conditions of the overlying beds is well illustrated by two examples in England. "At Sunderland, where the measures contain 50 per cent of hard-rock beds, seams at a depth of from 1,400 to 1,800 feet have been worked for seventy years without reference to the surface. On the other hand, in the Midland and South Yorkshire coal fields, where the cover is composed largely of soft shales, the effect of workings at as much as 2,000 feet is appreciable on the surface."*

Investigations of the thickness and physical character of each overlying bed are fundamentally necessary to the accurate study of subsidence in any district. Much of the data as to the behavior of various strata that can be secured will be at best only relative. However, the more data that can be secured the fewer will be the variables with which the investigator must deal.

Practically every theory of subsidence which has been advanced, when analyzed, involves some fundamental principle of mechanics. The beds may be subject to tension, compression, bending, or shear. Samples

*Eng. and Min. Jour., Vol. 84, p. 196, 1907.

of the various rocks may be tested in the laboratory in order to secure data to be used in the study of each problem. The great difficulty of obtaining specimens which will be representative of the section under investigation is largely responsible for the scarcity of data along certain lines, notably those concerning the strength of rock in tension and in bending.

Data on the strength of the rocks that are of importance in the study of subsidence have been collected and published by Bunting.*

"Numerous tests of various stones have proved that sandstones take permanent sets for the smallest loads, whereas granite and limestones are nearly perfectly elastic. It has also been proved by tests on various stones that the modulus of elasticity in compression is practically the same as in cross bending, but no fixed relation has been determined of the compressive, tensile, or shearing strength of the various kinds of stone.

"The shearing strength of sandstones and slates per square inch is generally slightly in excess of the modulus of rupture, and the compressive strength of various stones is variable and of comparatively little consequence here, as the compressive strength of even the lightest sandstone ranges from 4,000 to 6,000 lbs. per sq. inch."

The moduli of rupture of various kinds of stone as given by a number of authorities are shown in Table 6.

"Safe unit stresses for various stones have been given by many authorities. Below are given the stresses in pounds per square inch recommended by W. J. Douglas as illustrative of possibly a fair average of such values:

SAFE UNIT STRESSES FOR STONE.

	Compressive Lb. per Sq. In.	Shear Lb. per Sq. In.	Tension Lb. per Sq. In.
Blue stone flagging.....	1,500		200
Granite	1,020	200	150
Limestone	800	150	125
Sandstone	700	150	75

"It is to be observed, in the case of sandstone, that a safe tensile strength of 75 lbs. and a shearing strength of 150 pounds per square inch are given. Now, in consideration of the fact that the modulus of rupture is invariably in excess of the tensile strength, also that the resistance to shear slightly exceeds the modulus of rupture, a value of 100

*Bunting, D. "The Limits of Mining Under Heavy Wash." Amer. Inst. Min. Engrs., Bul. No. 97, p. 1, 1914.

pounds per square inch for the modulus of rupture of sandstone would be consistent. . . .

"When sandstones and slates, which generally overlie the coal veins, are considered as beams or slabs spanning mine openings for the support of overlying strata or other superimposed load, their transverse strength

TABLE 6.
MODULI OF RUPTURE OF STONES.

	Maximum	Minimum	Average	Authority
Blue stone flagging.....	4,511	360	2,700	Baker
Slate	9,000	1,800	5,400	Baker
Slate	11,230	7,425		Arsenal tests, 1902
Slate			8,480	Merriman
Granite	2,700	900	1,800	Baker
Granite			1,754	Merrill
Granite			2,610	Arsenal tests, 1907
Granite			1,667	Arsenal tests, 1905
Granite			1,365	Bauschinger
Granite			1,194	Bauschinger
Glass	3,500			Church
Glass	4,132			Fairbairn
Sandstone			1,576	Technology Quarterly
Sandstone			1,200	Merriman
Sandstone	1,273	655		Arsenal tests, 1895
Sandstone	2,243	1,500		Arsenal tests, 1895
Sandstone	2,340	576	1,260	Baker
Sandstone (variegated)			469	Bauschinger
Sandstone (variegated)			718	Bauschinger
Sandstone (variegated)			1,109	Bauschinger
Sandstone (variegated)			341	Bauschinger
Sandstone (carboniferous)			483	Bauschinger
Sandstone (slaty)			249	Bauschinger
Sandstone (slaty)			135	Bauschinger
Sandstone (green)			156	Bauschinger
Sandstone (cretaceous)			597	Bauschinger
Sandstone (cretaceous)			967	Bauschinger
Gray stone	2,200			Kent
Light stone	1,170			Kent
Stone			2,000	Merriman
Quartz conglomerate			654	Merriman

From the results of tests as given in Table 6, the average moduli of rupture of the various stones are as follows:

	Pounds per Square Inch
Blue stone flagging.....	2,700
Slate	7,736
Granite	1,681
Sandstone	806

is of first importance. The ability of such material to serve as a beam depends upon its tensile strength, since that is always less than its compressive strength." The action of the atmosphere and of water upon

rocks which have previously been protected from these natural agents occasionally reduces the strength of rocks.

TABLE 7.
SPECIFIC GRAVITY OF ROCKS.*

Rocks	Average Specific Gravity	Lb. Wt. per Cu. Ft.	No. of Cu. Ft. per Ton 2,000 Lbs.
Quartz	2.6	162.1	12.3
Andisite	2.9	181.0	11.1
Basalt	2.9	181.0	11.1
Diabase	3.0	187.0	10.6
Diarite	3.0	187.0	10.6
Granite	2.7	168.0	11.9
Limestone	2.7	168.0	11.9
Porphyry	2.7	170.0	11.8
Rhyolite	2.4	149.6	13.4
Sandstone	2.4	149.6	13.4
Schist	2.7	168.0	11.9
Shale	2.6	162.1	12.3

As previously noted, structural features are of great importance and the application of theories and rules will serve only as an indication of tendencies and possibilities. If the various rocks and strata were uniformly homogeneous the problem would be greatly simplified.

Natural processes may give rise to conditions which result in surface subsidence. Possibly the most comparable examples of subsidence due to natural agencies are those of surface sinks which result from the removal of portions of the supporting minerals by natural agencies.

Numerous examples of sink-holes and caves have been noted in the salt districts of Europe and in areas underlaid by calcareous materials which may be dissolved in part by underground waters.† In the United States similar phenomena have been noted. The sink-holes of the Ozark plateau have been studied in Missouri‡ and the information available indicates that they have been caused either by the caving of the roof over solution basins in limestone beds, or by the enlargement through solution of joints leading from the surface to an underground channel. It is probable that the larger sinks are the result of the former cause. Usually these sinks vary in diameter from 100 to 300 feet, although single sinks are known to include as much as 150 acres.

In Illinois, near Millstadt, in the Waterloo Quadrangle numerous sinks have resulted from solution cavities in limestone beds lying at shallow depth.

Apparently the same forces which act during the subsidence of the

*Herzig, C. S. "Mine Sampling and Valuing," p. 139, San Francisco, 1914.

†Woodward, H. B. "Geology of Soils and Substrata." London, 1912. Quotes Darwin on p. 64, reference to Cheshire salt district, p. 67.

‡Crane, G. W. "Iron Ores of Missouri." Missouri Bureau of Geology and Mines. Vol. 10, 2d ser., p. 34. Lee, Wallace, "Geology of the Rolla Quadrangle." Vol. 12, 2d ser., ch. VIII.

surface over these cavities caused by nature also cause subsidence over mine workings. It has been suggested that the fissure systems in volcanic areas have resulted from vertical movement or settling due to the transfer of material by volcanic action to the surface; the resulting cavity having probably been closed, in part at least, by the subsequent settling of the surface under the load of extrusive material.* The dropping of a block of the earth's crust tends to produce normal faults, and it may be appropriate to consult the authorities on structural geology regarding the observations which have been made upon faults and fractures which apparently have resulted from forces and processes somewhat similar to those which characterize subsidence due to mining.

As will be noted later, the investigator of subsidence desires to learn among other things how the strata bend and break when subjected to various forces, and in what direction fracture will occur when various forces act. He desires to learn how rapidly the deformation of rocks occurs and to what depths mining openings may be carried with safety. In the laboratory the angle of break of various rocks may be measured and many other data may be obtained, but the investigator requires also data based upon larger volumes of materials, greater and more slowly acting forces, and conditions more nearly approximating those which result from mining operations.

1. *Cleavage*.—"The planes of cleavage, incipient or pronounced, existing in the overlying roof strata may strike in the same general direction as the planes of cleavage existing in the coal below. The importance of this principle and the necessity of its acceptance justify a reference by way of proof to the natural philosophy of the case. According to geological theory, the cleavage in the coal and in the roof strata was produced by the action of the same force. Assuming that in a given case the planes of cleavage are vertical, the theory is that some force, acting laterally and at right angles to what are now planes of cleavage, was the cause of such cleavage being created in the strata. Such a lateral force is supplied by the shrinkage of the earth's crust. This force, acting with immense energy on the particles of matter in the strata and subjecting them to enormous lateral compression, obliged such particles so to arrange themselves that their longer axes finally lay at right angles to the line of action of the compressing force. The planes of cleavage are thus defined as the planes in which the particles of matter now extend their longer axes."†

*Lindgren, W. "Mineral Deposits." P. 136.

†Halbaum, H. W. G. "The Action, Influence and Control of the Roof in Longwall Workings." Trans. Inst. Min. Eng., Vol. 27, p. 214, 1903.

2. *Fractures.*—The subject of fractures has been discussed in various works on geology, notably in the recent work of Leith.* “Under tension fractures tend to develop in planes normal to the maximum stress. Tension fractures may develop when a mass is deformed by shearing. Under compressive stresses, fractures tend to develop above the planes of maximum shear, which are inclined to the direction of principal stresses; but the degree of inclination and the direction of dip of the planes away from the direction of maximum stress vary.”† Joints in rocks may be due to tension or to compression. Faults, which are “fractures along which there has been some relative displacement of the rock,”‡ may be regarded as the result of tension or of compression. A “gravity” or “normal” fault is generally the result of tension while compression causes “thrust” or “reverse” faults.

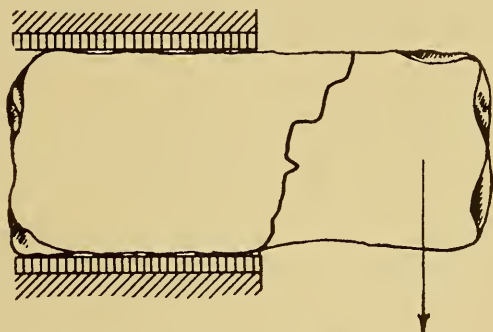


FIG. 22. ANGLE OF FRACTURE OF STONE.

The angle of fracture of rocks under stress has been noted and measured in the field and in the laboratory. Daubrée carried on extensive experiments in 1879 to show the effect of tension and compression.¶ Experiments made upon wax and resin prisms showed that compression causes rupture along a plane at an angle of 45 degrees to the line of force. If there has been preliminary deformation, the angle will be greater than 45 degrees.

Fayol tested pieces of sandstone and shale, as shown in Fig. 22, to discover the angle of fracture when the test piece is held firmly by one end and subjected to a steady and increasing pressure applied upon the projecting portion.

Leith states that data given in United States Geological Survey

*Leith, C. K. "Structural Geology." 1913.

†In subsidence following the advance of longwall mining strata may first be subject to tension and later to compression.

‡Leith. P. 81.

¶Daubrée, A. "Études Synthétiques de Géologie Expérimentale." P. 179.

Folios show an average dip of 78 degrees for normal fault planes and 36 degrees for reverse fault planes. Faults noted in Illinois have dips ranging from 35 degrees to 75 degrees, the majority, however, approximate 55 degrees.

Lindgren observes* that veins may dip at any angle but "veins dipping 50 degrees to 80 degrees are most common."

Stevens has formulated a law of fissures.† "In a homogeneous mass under pressure, slipping tends to take place only along those planes on which the ratio of tangential stress to direct stress is equal to the coefficient of friction of the material sliding on itself. If the axis of greatest principal stress is vertical, the displacement along the fissure will be that of a normal fault, and the dip of the normal fault which is most favorable to slipping will be 66 degrees. Similarly, when the axis of greatest principal stress is horizontal, the displacement along the fissures is that of a reverse fault, and the dip most favorable to slipping is 24 degrees."

Spencer has studied in the field the veins of southeastern Alaska.‡ There is a systematic arrangement of veinlets in two main sets standing at right angles to each other and dipping in opposite directions. Becker¶ concluded that the fracture had been produced through compressive shearing stresses which were caused by nearly tangential forces acting in a direction normal to the strike of the two sets of fractures. Spencer supports the theory that these fractures were caused by compressive thrust but questions the statement that the thrust was the result of tangential compression. He developed the theory that the general fissuring was a result of "gravitative adjustment in the rock-masses, tending to restore internal equilibrium disturbed during the uplifts which are known to have taken place." A broad mountainous zone rises about 5,000 feet above the interior plateau and 15,000 feet above the plateau bordering the Pacific Ocean. "Standing so far above the neighboring earth blocks, it seems that in this great orographic mass there must even now exist a tendency to bulge toward the unrestrained sides. If so, conditions are favorable for the opening of fractures at a depth dependent upon the crushing strength of the rocks which compose the great mountainous mass."

*Lindgren "Mineral Deposits." P. 151.

†Stevens, B. "The Laws of Fissures." Trans. Amer. Inst. Min. Engrs., Vol. 40, p. 475, 1909.

‡Spencer, A. C. "The Origin of Vein-Filled Openings in Southeastern Alaska." Trans. Amer. Inst. Min. Engrs., Vol. 36, p. 581, 1906. "The Geology of the Treadwell Ore Deposits." Trans. Amer. Inst. Min. Engrs., Vol. 35, p. 507, 1905.

¶Becker, G. F. 18th Annual Report, U. S. Geol. Sur. Pt. III, pp. 7-86.

The discussion and theory of Spencer is of great interest in this connection as it develops the idea of the settling of a mass of rock under its own weight and when movement is less restrained in one direction than in another. It also emphasizes the question of angle of fracture and systems of fractures which will be referred to later.

The theory of a "dome of equilibrium" developed by Fayol suggests the question of the possibility of removing a layer or bed from the earth without disturbing the surface, owing to the sphericity of the earth. The question of the supporting power of the dome of the earth's crust has been studied by a number of eminent geologists. Chamberlain and Salisbury refer to each portion of the crust as "ideally" an arch or dome. When large areas like the continents are considered, it is the dome rather than the arch that is involved, and in this the thrust is ideally towards all parts of the periphery. According to Hoskins, a dome corresponding perfectly to the sphericity of the earth, formed of firm crystalline rock of the high crushing strength of 25,000 pounds to the square inch, and having a weight of 180 pounds to the cubic foot would, if unsupported below, sustain only $1/525$ of its own weight. This result is essentially independent of the extent of the earth's radius.*

The idea that extensive areas can be left entirely unsupported if the curvature of the arch corresponds to the sphericity of the earth is entirely unwarranted, judging from the calculations made and from the experience at many mines.

Various structural features must be noted in determining the cause, effect, and probability of subsidence following mining operations. Among the most important of these are the conformability of the overlying rocks, joints, cleavage, bedding planes, folds, faults, fissures, dikes, and intrusives.

In many mining districts there are heavy beds of surficial material which complicate the problem on account of the water they contain and because they are more or less fluid and have little supporting power. The lateral extent of subsidence is greater when the area is covered with such beds. This is due largely to the smaller sliding angles upon which beds of sand, earth, marl, and gravel will move.

EXPERIMENTS TO DETERMINE ROCK FRACTURE.

Many experiments have been carried on by eminent geologists in order to discover by work in the laboratory fundamental data upon

*Chamberlain, T. C., and Salisbury, R. D. "Geology." Vol. 1, p. 681.

which theories may be based and also to verify if possible, by artificial processes, theories accounting for conditions which may have been the results of complex forces and reactions.

As previously noted, numerous tests have been made to determine the strength of rocks and minerals under various conditions and various properties of rocks have been studied.* Among the most interesting experiments in addition to the tests of materials are the following: Fayol conducted elaborate tests of materials such as those which composed the beds overlying the Commentry Mine and by ingeniously constructed models attempted to measure the lateral and vertical extent of subsidence.† The work of Daubrée has been noted previously. Extensive experiments have been made also by Meade and by Pauleke. In America among the experiments which have attracted most attention are those described by Willis in "Mechanics of Appalachian Structure"; those by Adams and Coker on elastic constants, flowage, and the cubic compressibility of rocks; those by Becker on schistosity and slaty cleavage; and those by Hobbs on mountain formation.

Most of these experiments consider tangential pressures rather than vertical pressure. Very few of them develop conditions which approximate those which occur when the support of rock is removed.

*Consult the Bibliography, p. 180, for records of these experiments.

†See page 76.

CHAPTER III.

THEORIES OF SUBSIDENCE—GENERAL PRINCIPLES.

In this bulletin no attempt will be made to discuss theories of mechanics or derive formulæ applying to subsidence, but an effort will be made to state briefly the conditions that exist and to point out the fundamental and controlling factors in a study of the problem.

In order to study the reactions which may exist in the rocks overlying a mineral deposit, it will be necessary to make certain assumptions in order to arrive at some definite conclusions. For example, it must be assumed that the rock is uniformly of a known strength, that it is free from structural weaknesses, and that it exists in masses or beds whose extent, thickness, depth, and dip are known.

The principles of mechanics may be applied to various types of mine openings, notably: (1) The long narrow excavation which may be driven through massive or bedded rocks, or along the strike or the dip of bedded rocks, as tunnels, drifts, crosscuts, and entries. (2) Excavations of greater width, as rooms or stopes. (3) Excavations of great lateral extent, as those of a longwall coal mine, or sections of a pillar-and-room mine after the pillars have been drawn. In these various types of openings the fact must be recognized that maximum pressure may not always be due to a thrust acting vertically downward.

In order to simplify the problem it may be suggested that the rock and mineral overlying and surrounding the excavation be considered as forming one of the following:

- (1) A beam of rock lying horizontally or inclined and extending from pillar to pillar or column to column.
- (2) A cantilever supported by a pier of rock or mineral.
- (3) An arch or series of arches of equal or unequal spans.
- (4) A column or pier, either vertical or inclined, supporting (1), (2), or (3).
- (5) A dome of the earth's crust.

It should be noted further that when the roof is considered as acting as a beam it may be supported by piers of mineral, of noncoherent filling, of timber, or of masonry, resting upon a more or less yielding floor. With these explanatory statements, the various theories of subsidence that have been formulated will be considered.

HISTORICAL REVIEW OF THEORIES OF SUBSIDENCE.*

Belgian-French Theories.

Belgian engineers were among the first to make a scientific study of earth movements due to mining operations. In 1825 a commission investigating the cause of surface cracks about the city of Liege expressed the opinion that a distance of 300 feet between the mine workings and the surface is more than sufficient to protect the surface. Further disturbance of the surface raised the same questions in 1839. Another commission of mining engineers concluded that there would be no danger to buildings or wells from mining operations at a depth of 300 feet.†

Although credit for formulating the first theory of subsidence is usually given to the Belgian engineer, J. Gonot, it is claimed by L. Thir-

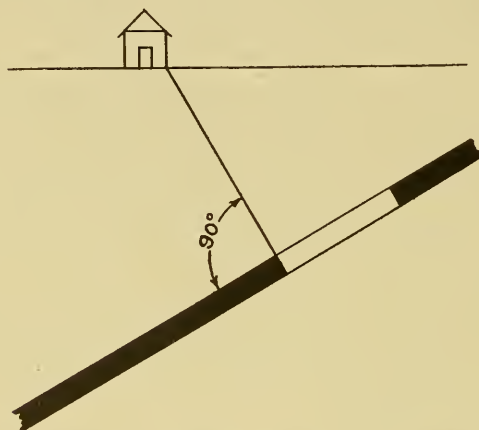


FIG. 23. DIAGRAM ILLUSTRATING THE "LAW OF THE NORMAL."

hart that the fundamental idea of the theory of the normal was first presented by the French engineer, Toillez, in 1838. Gonot studied surface subsidence in the vicinity of Liege in 1839 and formulated a theory which was published in 1858. He claimed that following the removal of coal the overlying strata would sink and the angle of fracture would be perpendicular to the plane of the coal bed. (Fig. 23.) This theory was later referred to as the "Law of the Normal." Mining operators in general and many engineers criticised this theory and, while many later writers accepted the principle as it applied to horizontal

*Fayol, H. "Sur les Mouvements de Terrain Provoqués par l'Exploitation des Mines." *Bul. Soc. Ind. Min.*, IIe ser., Vol. 14, p. 862; Kolbe, E., "Translocation der Deckgebirge durch Kohlenabbau," pp. 2-51, Essen, 1903.

†Vuillemin, E. and G. *Bul. Soc. Ind. Min.*, IIe ser., Vol. 14, p. 858, 1885.

and slightly dipping beds, various qualifications were suggested in regard to the angle of fracture of steeply inclined beds. (Fig. 24.) Gonot also held that the break extends through to the surface, irrespective of the depth of mining. He based this theory on observations he had made on subsidence at Liege. The Belgian engineer, Rucloux, who was appointed with Wellekens to investigate subsidence about Liege in 1858, called attention to the fact that Gonot's theory undoubtedly could not be applied to vertical and highly inclined beds. While many criticisms were offered, no new theory was presented. The commission held that the observed facts were sufficient to establish the principle that with solid beds of an ordinary thickness and at moderate depths exploitation by contiguous openings and successive fillings up to a considerable extent may be made without affecting the surface. Where the depths

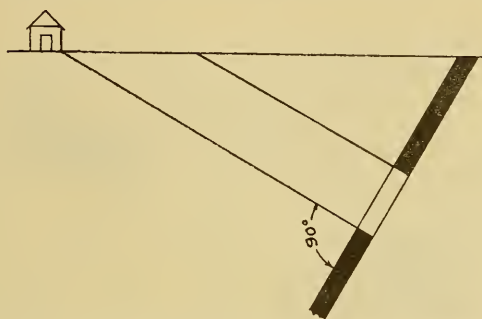


FIG. 24. THE "LAW OF THE NORMAL" NOT APPLICABLE TO STEEPLY DIPPING BEDS.

are slight, or when for one reason or another the beds lose their solidity, subsidence may be prevented by preserving pillars. The subsidences which are produced on account of the underground work generally follow vertical lines, but may deviate from these lines according to the direction of the beds, more often toward the lower side and often also toward the upper side.

In 1868 four engineers were commissioned by the Prussian Government to collect information on the question of the "influence that mine workings may have on surface building" in the coal fields of various countries. They found that at that time the majority of Belgian engineers believed that when the coal is entirely removed the most careful packing gives no guarantee against damage to surface building; that the packing only lessens the sinking; and that the surface may be protected by leaving pillars. In order to make this method effective only half the area of the coal seams must be removed.

In 1871 the Belgian engineer, G. Dumont, who had been appointed to make an investigation of conditions in and about Liege, made a careful study of the problem and submitted a voluminous report of 331 pages, in which he supported the fundamental idea of the "Law of the Normal" but limited its applicability to beds dipping not more than 68 degrees from the horizontal. This conclusion was based in part upon upwards of a thousand levels at various parts of the town. He* called attention to the direction and amount of the forces acting on the block of rock overlying the excavation. The broken pieces must fall into the excavation, and on highly inclined seams, according to Gonot's theory, the masses of broken rock would have to move toward the excavation on an angle less than the sliding angle. If $a-b$, in Fig. 25, represents the weight of the rock A-B, and this force is resolved into the forces $a-d$ and

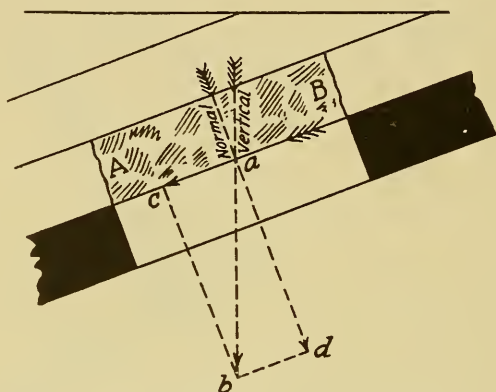


FIG. 25. FORCES ACTING ON ROCK IN AN INCLINED BED.

$a-c$, it is evident that, as the bed becomes steeper, the force corresponding to $a-d$ will become less and the force corresponding to $a-c$ greater. The tendency, then, will be to create a cavity vertically above the excavation rather than in a direction perpendicular to the plane of the bed.

Dumont held that the "inclination of the strata lessens the depth of the subsidence, but increases the area damaged. Timbering hinders the beds forming the roof of a seam from breaking, and therefore prevents the increase in their volume, which takes place when they break. It thus increases rather than diminishes the subsidence at the surface."†

*Dumont, G. "Des Affaisements du Sol Produits par l'Exploitation Houillère," Liege, 1871.

†Colliery Engineer, Vol. 11, p. 25, 1890-91.

The period during which the movement of the surface may continue is uncertain. In Belgium it extends generally over ten to twelve years but in certain instances has been known to continue twenty and even fifty years. The draining of old workings or the flooding of a mine may bring about fresh movements a long time after the original movement has ceased.

J. Callon, of the École des Mines, Paris, supported Gonot's theory but with some reservations.* He believed that when the coal bed is overlaid with unconformable beds, the angle of fracture will extend through each bed perpendicular to its plane of bedding. (Fig. 26.) He held that the amount of surface subsidence would depend on the compressibility of the material which fell into the excavation. In hard rocks a cavity narrowing upwards would be formed, while in soft rocks the cavity would be funnel-shaped.

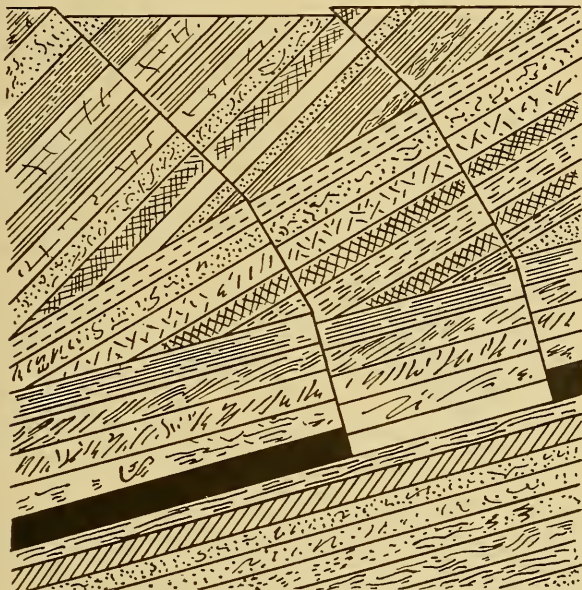


FIG. 26. FRACTURE NORMAL TO BEDDING PLANE.

The Colliery Owners' Association of Liege published a reply to Dumont in 1875.† The validity of Gonot's theory for beds of low dip was admitted, but his claim that the fracture would be normal to highly inclined seams was disputed. They argued that the fracture over the

*"Cours d'Exploitation des Mines," Vol. 2, p. 334, 1874.

†"Des Affaissements du Sol. Attributés à l'Exploitation Houillère," Liege, 1875.

workings would take place in a series of breaks approximately perpendicular to the bedding plane of each stratum, but that the force of gravity would cause the material to fall from the outcrop side of the excavation, causing the line of fracture to lie between the vertical and the perpendicular to the vein; while on the lower side of the excavation, each bed would tend to support the bed above and there would be an overhanging of slabs of rock toward the excavation. Thus the line of

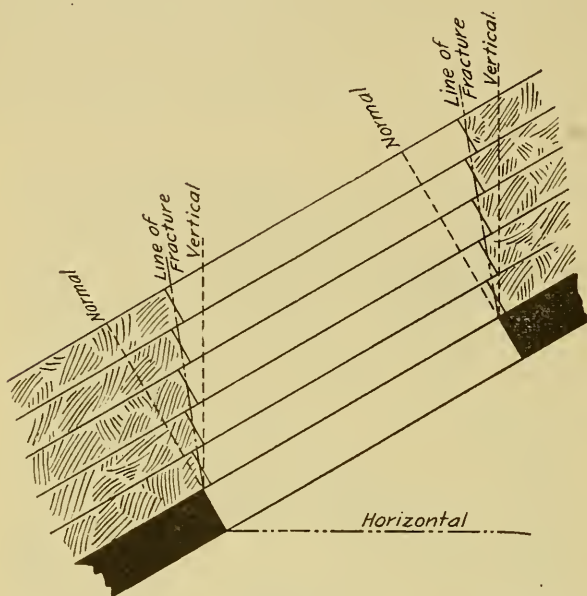


FIG. 27. LINE OF BREAK BETWEEN NORMAL AND VERTICAL.

fracture would be between the vertical and the normal to the bedding planes. (Fig. 27.) They also called attention to Coulomb's measurement of the angle of fracture by crushing. "The combination of this force producing crushing with that tending to break the bed by bending induces fracture along a line intermediate between the two directions, and such line goes further from the normal as the inclination of the strata increases."* On the whole the Colliery Owners' Association thought the Dumont's theory was unsatisfactory and often of no practical use and

*Hughes, H. W. "Textbook of Coal Mining," London, 1904.

that the only rule to follow was the examination of the special facts in each particular case.

M. Haton de la Goupilliere (1884), Professor of Mining at the École des Mines, Paris, held views similar to those of Callon. He pointed out the effect of the fallen material, which tends to check subsidence and in fact may stop it at a certain level. With longwall mining and filling he thought the movement would be almost independent of the depth. He held that it would be impossible to have the "Law of the Normal" completely verified in practice.*

The continued subsidence of the surface at Liege and the disagreement among engineers as to the theories of subsidence induced H. Fayol† to make observations of elevations at mines and to conduct laboratory experiments. He first summarized the contradictory opinions of the time as follows:‡

(1) Upon the extension of the movement upwards.

(a) The movement is transmitted to the surface whatever may be the depth of the workings.

(b) The surface is not affected when the workings exceed a certain depth.

(2) Upon the amplitude of the movements.

(a) Subsidence extends to the surface without sensible diminution.

(b) Movements become more and more feeble as they extend upwards.

(3) Upon the relative positions of the surface subsidence and of the mining excavation.

(a) Subsidence always takes place vertically above the workings.

(b) Subsidence is limited to an area bounded by lines drawn from the perimeter of the workings and perpendicular to the beds.

(c) Subsidence can not be referred to the excavation either by vertical lines or lines normal to the beds, but only by lines drawn at an angle of 45 degrees to the horizon, by the angle of repose of the ground, or by some other similar angle.

(4) Upon the influence of gobbing.

(a) The use of packing protects the surface effectually.

(b) Packing simply reduces the effect of subsidence.

*"Cours d'Exploitation des Mines," 1883.

†Director, Commentry and Montvicq Mines in France.

‡Bul. Soc. Ind. Min., IIe ser., Vol. 14, p. 805, 1885.

(c) Subsidence is greater with stowing than without it.

Fayol conducted a long series of investigations and experiments* and came to the conclusion that the movements of the ground are limited by a kind of dome which has for its base the area of the excavation and that their amplitude diminishes by degrees as they extend further away from the center of the area.

"Fayol's rule agrees with all the facts observed; absence of subsidence, more or less important subsidences, movements limited to the vertical above the perimeter of the excavations, those limited to the normal or to other inclinations, and so on. It has the disadvantage of being indefinite; but in a question which embraces so many elements, many of which are unknown or not well known, such as the nature of the rocks, the thickness of the beds, irregularities in geological structure, the action of water, etc., we cannot hope to arrive at absolutely accurate formulæ; we shall have accomplished much when we get to know very nearly the true form, the direction, and the relative amplitude of the subsidences, and are in a position to combat false ideas successfully."†

According to Fayol the disturbance of the strata is greatest over the center of the area excavated and it diminishes in amount toward the perimeter of the excavated area. As the vertical distance above the excavation increases, the amount of the movement decreases, and, if the workings are at great depth, there will be a depth beyond which the movement will cease. When graphically represented the limits of the movement are depicted by a dome; outside of this dome there can be no disturbance whatever. However, Fayol called attention to the possibility of movement if there should be a series of these domes in close proximity to each other, and to the effect of dip, rock structure, etc. upon the practical application of this theory. As a result of his experiments and observations, Fayol concluded:

(1) If excavations were stowed in a thoroughly tight and efficient manner with incompressible materials there would be no subsidence, but ordinary stowing is not done under these conditions, because the materials employed are all more or less compressible and the excavations are never perfectly filled up. When the roof settles the stowing resists feebly at first, after which the resistance rapidly increases and finally arrests the downward movement.

(2) The amplitude of the subsidence diminishes in proportion to

*See page 138.

†Galloway, W. "Subsidences Caused by the Workings in Mines," Proc. South Wales Inst. of Engrs., Vol. 20, p. 311, 1897.

the depth of the workings below the surface, the diminution being proportional to the increase of depth.

Leon Thiriart in 1912 called attention* to the theory of Banneux, which Thiriart called the "Law of the Tangent." Thiriart's theory is a modification of Banneux's, and Banneux's theory resembles that of Hausse.† The bending moment is considered for each bed successively, beginning with the one immediately overlying the coal. By elaborate calculation, based on observations of subsidence, formulæ are derived by which a table showing the angle of break for various dips has been compiled.

German Theories.

A. Schulz, one of the first German engineers to study surface subsidence due to mining operations, in 1867 published his ideas on the

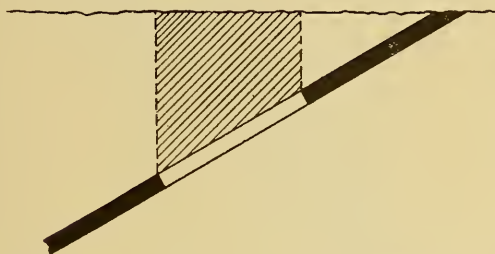


FIG. 28. VERTICAL FRACTURE OF DIPPING BEDS OF SHALE.

angle of fracture and the size of pillars necessary to protect objects on the surface.‡

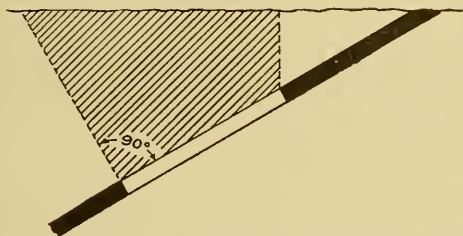


FIG. 29. SCHULZ'S IDEA OF FRACTURE OF SANDSTONE BEDS.

He criticised Gonot's "Law of the Normal" and held that in dipping beds of shale the fracture will occur along vertical planes (Fig. 28), while in sandstone the fracture on the dip side will approach the nor-

*Thiriart Leon "Les Affaissements du Sol Produits par l'Exploitation Houillère," Ann. des Mines de Belgique, Vol. 17, p. 3, Bruxelles, 1912.

†See page 97.

‡Schulz, A. "Investigations on the Dimensions of the Safety Pillars for the Saarbuck Coal Mining Industry and on the Angle of Fracture at Which the Strata Settle Into Worked-Out Rooms." Zeit. für B., H., u. S.-W., 1867.

mal, but on the outcrop side it will be vertical. He held in general that the fracture would occur between the vertical and the normal to the bed.

During the same year that Schulz published his paper on the angle of fracture, Mining Assessor von Sparre published a criticism of Gonot's theory.* He held ideas similar to those of Schulz; namely, that the fracture will occur between the vertical and the normal. He suggested the consideration of the separate beds and showed that for each bed the fracture would occur between the vertical and the normal on both sides of the excavation in a dipping coal seam. As shown in Fig. 30, the bounding planes of the break will be not *ab* and *lh* nor *ac* and *hm* but midway between.

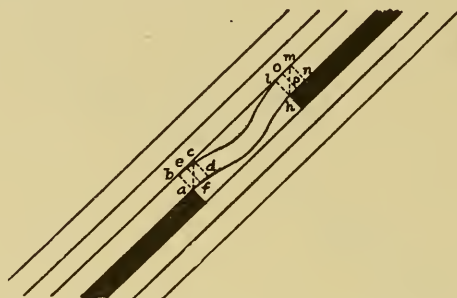


FIG. 30. FRACTURE IN DIPPING BEDS ACCORDING TO VON SPARRE.

Von Dechen called attention in 1866 to the importance of studying the part played in subsidence by the heavy marl beds overlying the coal measures. Some engineers, in fact, held that subsidence was due entirely to the unwatering and drying of these marl beds. Von Dechen noted also that the "Law of the Normal" could not be applied to very steeply inclined or vertical beds.†

In 1894 the project of constructing a canal between Herne and Ruhrart caused an investigation of the stability of the surface over which it was proposed to build the canal. A survey of conditions in the Dortmund district was made by the Board of Mines of Dortmund; levels were run and maps were made, and a very complete report was submitted in 1897.‡ It was concluded from observation on the West-

*Von Sparre, J. "On the Angle of Fracture of Strata of the Coal Measures," Glückauf, 1867.

†Von Dechen, H. "Opinion on the Surface Subsidence in and About the City of Essen," Manuscript, 1869.

‡"On the Influence of Coal Mines Under Marl Capping Upon the Earth's Surface in the Dortmund District." Zeitschrift für B., H., u. S.-W., Vol. 46, pp. 372-392, 1897.

phalian mines and for the conditions of that district that there would be no "harmless depth." The lateral extent of subsidence was found to increase greatly with thickness of the marl covering. It was noted that careful filling of the mine workings will greatly reduce the amount of vertical subsidence but will not affect greatly the lateral extent, in fact, in some instances it was found that with filling the lateral extent of subsidence is greater than when no filling whatever is used.

From the data collected an effort was made to determine the angle of fracture in rock and also the angle at which the limiting plane of subsidence of the marl extends from bedrock to the surface. In rock dipping not more than 15 degrees, the angle of fracture was found to

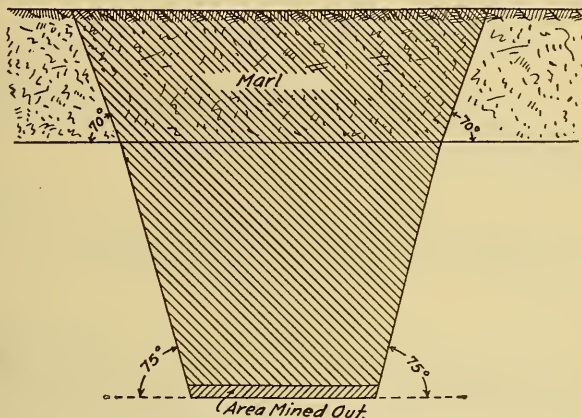


FIG. 31. SUBSIDENCE BEYOND ANGLE OF BREAK ACCORDING TO WACHSMANN.

be about 75 degrees, measured from the horizontal (Fig. 31), while in steep seams the angle approaches the natural slope, generally not less than 55 degrees.

For dips up to 65 degrees the vertical amount of subsidence may be found by the formula:

$$S = f. m. \cos a$$

in which S = vertical amount of subsidence,

m = thickness of coal worked,

a = angle of dip of coal seam.

f is a coefficient whose maximum values are as follows, if filling is complete:

0.40 for dips 0—10°,

0.30 for dips 10—35°,

0.25 for dips over 35°.

When no filling is used f may be as much as 0.80.

Wachsmann* held that when an underlying coal bed is mined, the lowermost strata collapse, the next higher strata sink and crack if the

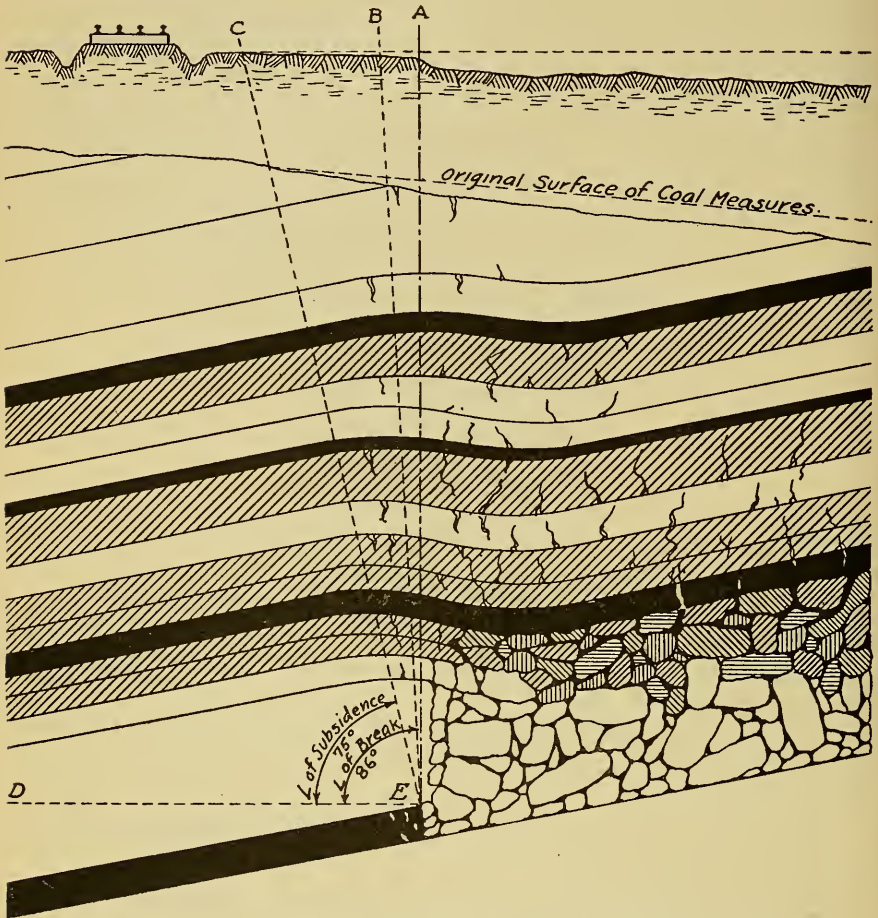


FIG. 32. ANGLES OF FRACTURE IN ROCK AND OF SUBSIDENCE IN MARL.

excavation is of sufficient extent, while the uppermost strata sink without breaking or cracking. There is subsidence beyond the actual angle of break, as shown in Fig. 32.

In 1885 Mining Engineer R. Hausse published some observations

*Über die Einwirkung des Oberschlesischen Steinkohlenbergbaues auf die Oberfläche. Zeit. f. Obersches B.- u., Hüttenmannischen Verein, p. 313, 1900.

on the angle of break.* Owing to the great interest in the problem of subsidence, Hausse directed his entire time to the scientific investigation of various phases of the problem in Germany, particularly in Saxony, and published the results of his work in 1907.† In his preliminary

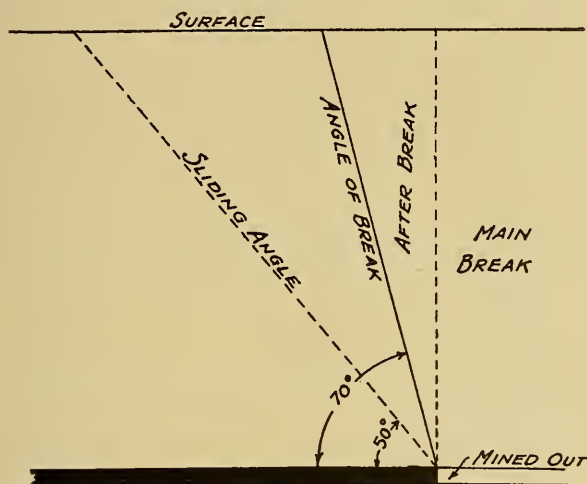


FIG. 33. HAUSSE'S THEORY; ANGLE OF BREAK BETWEEN VERTICAL AND SLIDING ANGLE.

chapter Hausse discusses the behavior of rocks over excavated areas and distinguishes between the "first" or "main break" and the "afterbreak." Over a horizontal bed he found that the main fracture is vertical and that the after break extends upward over the unmined coal. He pointed out that after the first falls occur, filling the cavity, the overlying strata subside, compressing the fallen material as filling is compressed by the roof in longwall mining. He stated that the plane of fracture lies between the vertical and the plane of sliding for the rock in question, and generally the plane of fracture will bisect the angle between the two limiting planes (Fig. 33). He discussed the relation between the directions of the main break, the after break, and the angle of inclination of the seam, as follows:

The angle of main fracture on the upper side $= 90 \frac{a}{2}$, in which

a = angle of dip. The angle of after break is assumed to be either con-

*Hausse, R. "Von der Niedergehen des Gebirges beim Kohlenbergbau und den damit Zusammenhängenden Boden- und Gebäudesenkungen," Ziet für das Berg-Hütten und Salinenwesen, pp. 324-446, 1907.

†Hausse, R. "Beitrag zur Bruch Theorie; Ehrfahrungen über Bodensenkungen und Gebirgsdruckwirkungen," Jahrbuch für das Berg-und Hüttenwesen in Sachsen, 1885.

stant or equal to 20° , or it is taken as decreasing from 20° to 10° in proportion to the increase of the dip from 0° to 45° . In a series of tables, Hausse presents the angle of fracture for various dips, making certain assumptions. He determined coefficients of increase of volume for mining with filling and without filling on the basis of observations made in the Royal Colliery at Plauen under the Dresden-Tharandt Government Railway. Where there was no filling, the coefficient was found to be 0.01 and with filling the coefficient was found to be 0.002.

Starting from Rziha's assumption of sliding angles, Trompeter* determined by the use of Hausse's observations the breaking zone with

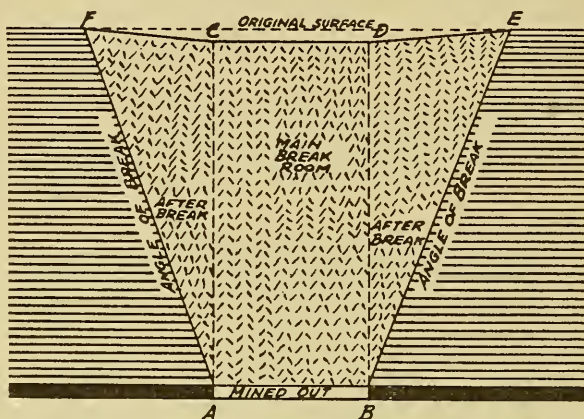


FIG. 34. "MAIN BREAK" AND "AFTER BREAK." (HAUSSE.)

regard to the expansive power or increase in volume of broken rock. From his experience he found this for the Rhenish-Westphalian coal district to be an increase of 12 meters for every 100 meters in depth.

Puschmann has described the subsequent working of overlying seams in the coal district of Upper Silesia.†

According to many engineers, the unwatering of the surficial material has been the immediate and sole cause of surface subsidence. Those who hold this opinion claim that surface movement has resulted from the sinking of shafts and the driving of boreholes alone and without any actual removal of the mineral deposit. There is, however, a wide difference of opinion on this matter. Von Dechen held that the

*Trompeter, W. H. "Die Expansivkraft im Gestein als Hauptursache der Bewegung des den Bergbau Umgehenden Gebirges," Oestreichische Zeit., für Berg-und Hüttenwesen, 1899.

†Puschmann Über den Nachträglichen Abbau Hangender Flöze beim Oberschlesischen Steinkohlenbergbau," Zeit. für d. B., H., u. S.-W., Vol. 58, p. 887, 1910.

subsidences* near Essen in 1866 and 1868 were caused by the partial drying of the marl and green sand overlying the coal measures. Graff† demonstrated by tests that drainage does not cause any change of volume of sand or quicksand and held that, when the water does not carry away any solids, there can be no subsidence resulting simply from unwatering.

The principal value of the work of the Germans has been in determining the angle of break, extent of surface subsidence relative to mine workings, and the practical coefficients of increase in the volume of the measures overlying the mined area, as these measures sink into the worked out area. Data upon these phases of subsidence have been collected in a systematic and accurate manner during a long period of years by engineers in the coal mining districts of Germany.

Austrian Theories.

Subsidence of surface due to mining operations has attracted considerable attention in the Ostrau-Karwin coal region in Austria. Ministerial regulations controlling the removal of coal under railways were formulated and became effective January 2, 1859. In 1876 Mining Director W. Jicinsky‡ published a treatise on subsidence.

The question of the applicability of the government regulations for the protection of railroads to the railroads leading to the mines and serving the mines almost exclusively aroused discussion in the late 70's and the early 80's. The Board of Mines of Olmutz framed a regulation to modify the existing regulations so that they would not apply so strictly to mining railroads. F. Rziha, Professor of Railroad Construction at the Technical School of Vienna, was engaged to advise on the proposed regulation. He expressed his opinion on this question, formulated a theory of subsidence, and presented regulations to govern the exploitation of the coal under the mining railway.§ Rziha formulated a theory covering (1) the direction of fracture and (2) the amount of the vertical sinking of the earth, a brief statement of which follows:

(1) The Direction of Fracture. When rock is undercut, there is a tendency for it to fall or sink in proportion as gravity exceeds cohesion. The action may be falling or tearing, or both. He distinguished what he

*Goldreich. Die Theorieder Bodensenkungen in Kohlengebieten, p. 20.

†Graff "Verursacht der Bergbau Bodensenkungen durch die Entwässerung Wassere-führender diluvialer Gebirgsschichten," Glückauf, 1901.

‡Jicinsky, W. "The Subsidence and Breaks of the Surface in Consequence of Coal Mining," 1876. Published later as a monograph of the Ostrau-Karwin coal district, 1884, "The Effects of Coal Mining on the Surface."

§Oest. Zeit. f. B.-, u. H.-W., Vol. 29, 1881, and Vol. 30, 1882.

called a "falling space" and surrounding it, more or less concentrically, a "friability" or "tearing space." He found the falling space to approximate the shape of a paraboloid. First the rock becomes loosened and afterwards falls when gravity exceeds cohesion. In time, limited by the structure of the rock, a dome-shaped space *abc* is formed (Fig. 35),

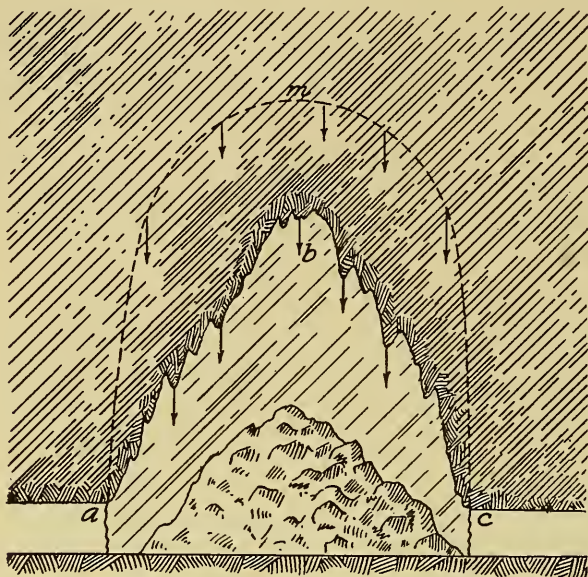


FIG. 35. "ZONE OF FALLING" AND "ZONE OF TEARING."

working laterally and vertically from the center of disturbance. Outside this space and more or less concentric is the dome of tearing, indicated by the dotted line *amc*. If the tearing sphere extends to the surface, it will cause surface disturbance within the area bounded by *mn* (Fig. 36), and the overhanging wall will gradually change its slope, depending upon the lateral extent and degree of the tearing and the relation between gravity and cohesion (see Fig. 37). Rziha thought that the stratification of the beds did not have much effect upon the angle of break. It may be noted that he did not make a detailed study of the subsided area in the field. The Mining and Metallurgical Society of M-Ostrau held that actual subsidence in the Ostrau-Karwin district did not conform to Rziha's theory.

(2) Vertical Movement. Rziha treated this subject under two headings: (a) The collapse into the underground excavation, and (b) the unwatering of the roof, causing a decrease in volume. He held

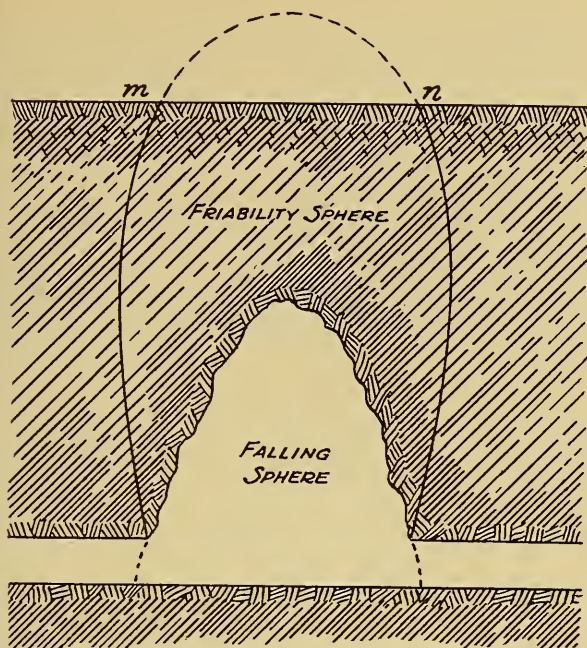


FIG. 36. "ZONE OF TEARING" EXTENDED TO SURFACE.

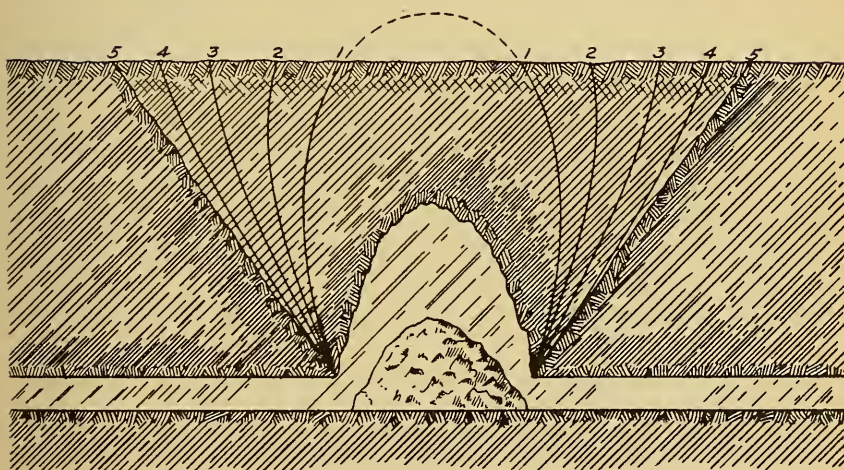


FIG. 37. SUBSIDENCE OUTSIDE UNDERMINED AREA.

that when mining is carried on at a great depth there may be no disturbance of the surface, and attempted to determine this depth by the formula:

$$h = \frac{M}{a}$$

in which h = harmless depth,

a = coefficient of increase of volume,

M = vertical seam thickness.

When pillars are left, he assumed that they will reduce subsidence, and the formula used to determine the harmless depth is:

$$h = \frac{M B}{a}$$

in which B is a coefficient for pillars and the filling, varying from 0.50 to 0.60. Rziha presented coefficients for the increase in volume of six kinds of rock. Goldreich* is of the opinion that an additional factor, overlooked by Rziha, is height of excavation. The less the height, the greater is the probability that the overlying rock will sink without complete or extensive crushing. Moreover, the completeness of the compression of the packing under the roof weight depends somewhat upon the height and extent of the excavation. These coefficients were not secured by mine investigation.

Goldreich criticises Rziha's suggestion of leaving pillars, and states: "If Rziha had been acquainted with the shape of the surface depressions, he would never have recommended the working methods given in his report." Objection is made also to the idea of a harmless depth as it "logically implies there ought to be the possibility of creating cavities of unlimited size beneath the harmless depth without giving rise to land movements on the surface." The Mining and Metallurgical Society of Ostrau (Moravia) to which organization Rziha's regulation was submitted, published its opinion in 1882.† Observations over a period of thirty years in the Austrian coal fields lead Goldreich to support Jicinsky's statement that no movement occurs if the water does not carry away from the volume under observation any solids mechanically or in solution.

Goldreich states‡ that the efforts to study theoretically the causes of subsidence in Austria date from the year, 1882. The Committee of the Mining and Metallurgical Society of Ostrau, Moravia, consisting of W. Jicinsky, J. Mayer, and von Wurzman, attacked Rziha's theory on the basis of observations in the Ostrau-Karwin district. Rziha failed to

*Goldreich Op. cit., p. 53.

†Oestrr. Zeit. für B.-, u. H.-W., Vol. 30, 1882.

‡Goldreich Op. cit., p. 45.

consider the probability of the overlying rock bending and sagging without breaking to fill the worked-out space, as shown in Fig. 38. In the event that sagging takes place, the increase in volume will be much smaller than that figured by Rziha, who assumed a breaking up of rock. (Fig. 39.)

The Committee preferred to use the term "undangerous depth" in addition to Rziha's phrase "harmless depth" for those depths at which

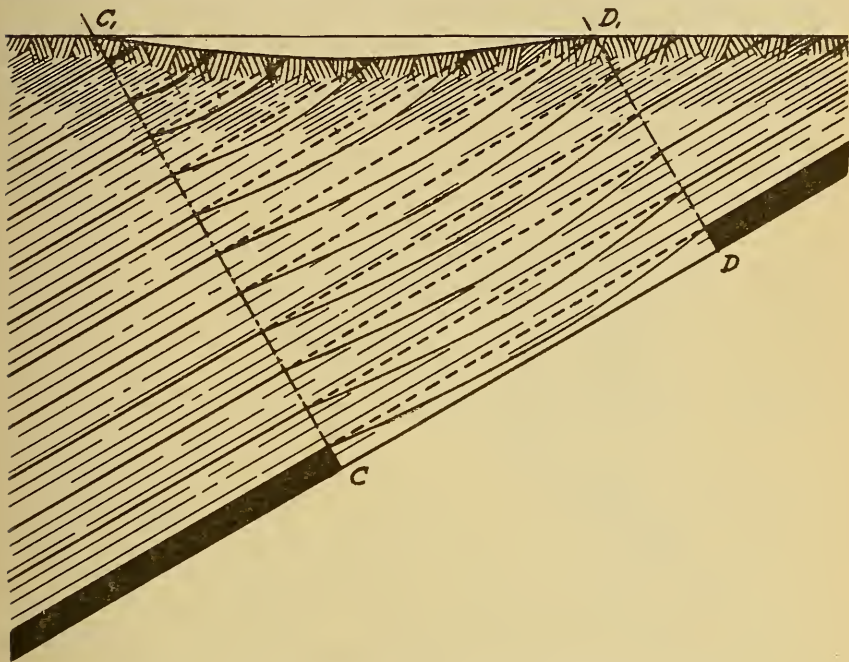


FIG. 38. LARGE SUBSIDENCE IN CASE OF BENDING OF ROCK.

mining will produce a gradual subsidence of the surface without damage to small objects or structures.

Jicinsky, a member of the committee, found the average increase in the volume of the rock of the coal measures and calculated the harmless depth from the maximum surface subsidence observed, according to the formula:

$$s = t + m - at$$

$$\text{or } a = 1 + \frac{m-s}{t}$$

in which

s = surface subsidence,

t = thickness of coal rock exclusive of the coal bed,

m = thickness of coal bed,

a = average coefficient of increase of volume of the coal rock considered as a whole.

The term "coal rock" means the bed of rock overlying the coal which is broken in the course of subsidence, with consequent increase of volume. If s is made 0, the formula shows the thickness of overlying rock, exclusive of the marl, necessary to prevent surface subsidence. The entire mass of material above the coal rock is believed to settle without increase in volume and is not taken into account in the formula.

The average coefficient of increase of volume of the coal rock is found to be 0.01 (i. e., $a = 1.01$) for several cases of subsidence indicated by Jicinsky. His conclusion is then, as expressed by the formula, that the surface subsidence is equal to the thickness of the bed taken out

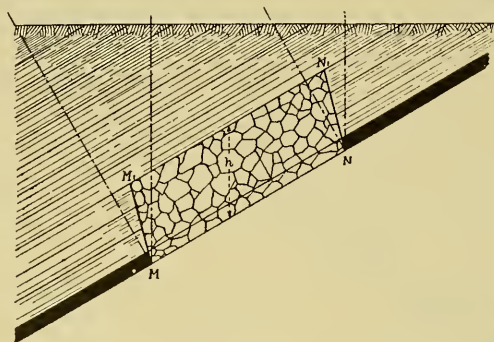


FIG. 39. SMALL SUBSIDENCE IN CASE OF BREAKING OF ROCK.

minus 0.01 of the thickness of the overlying rock which is shattered by movement.

The Committee also noted the vertical amount of subsidence and the duration of the movement, and special protective devices for shallow depths were considered. It was estimated that filling was compressed to 0.6 of the thickness of the bed so that in determining the harmless depth only 0.4 of thickness of the coal bed should be used in the formula. The committee agreed that "the interests of national economy demand that the leaving of coal pillars shall be prescribed in the rarest cases" and "the cost of the surface objects to be protected is to be compared with the coal losses." The Committee prepared regulations for coal mining under the mining railways, and these later became the basis of the regulations adopted.

Jicinsky's* monograph on subsidence appeared in 1884 and contained a complete statement of the principles accepted at that time by the leading Austrian engineers, and in it he formulated the following principles:

"The subsidence depends: (1) on the thickness of the seam, (2) on the dip of the seam, (3) on the depth of the mine, and (4) on the constitution of the roof rock of the seam. It may be regarded as a rule: (a) That the depth of the land subsidence is directly proportioned to the thickness of the seam, and the extension of subsidence is in direct proportion to the mined area; (b) that the depth of the surface subsidence increases with the dip of the seam, whereas its extension decreases, and for this reason in vertical seams the land subsidence is deep but manifests itself in the form of a kettle-shaped pit."

Jicinsky held further: "During every collapse of a solid rock there takes place a piling up of the broken masses. It follows that such a break causes an increase of volume and at a certain height in consequence of this increase in volume the entire empty space is so filled that no further after-break is possible; hence the effects of the break upon the surface must decrease with increasing depth. Every rock, even every single rock stratum, has its own hardness and toughness, and for this reason not all the kinds of rock behave in the same way during their collapse."

The subsidence formula of Jicinsky can be used to determine the harmless depth. Jicinsky discussed this point at length. He also made a study of the direction of fracture in the overlying rocks, and held that along the strike fracture is always normal to the coal bed. He objected to Gonot's and Schulz's theories for fracture on the dip and held that the angle was midway between the normal and the vertical. His views have been supported by 80 per cent of his observations.

He also considered in his monograph the amount of surface subsidence resulting from the mining of several superimposed seams, the various stages of rock movements, and safety pillars. In his opinion arbitrary rules cannot be used for determining the size of pillars, but each case must be studied separately and figured according to the local conditions.

C. Balling made a study of the angle of fracture in the northwest Bohemian brown coal basin and found that, for a depth of not more

*Jicinsky, W. "The Influences of Coal Mining Upon the Surface." Monograph of the Ostrau-Karwin Coal District, 1884.

than 300 feet and a dip of 8 degrees, the angle varied from 68 to 74 degrees.*

Chief Mine Inspector Anton Padour also made a report upon subsidence in the northwest Bohemian brown coal district.† He noted the vertical amount of subsidence and found the following relation:

$$H = 46 \sqrt[3]{h^2}$$

in which

H = vertical height of subsidence,

h = height of excavation.

He found that in the Bruch district, where the covering is firm marl, the angle of fracture is as follows:

(1) Thickness of capping from 1,000 to 1,200 feet, angle of seam from 0 to 8 degrees.

(a) towards the dip, angle varies from 72 to 69 degrees.

(b) towards the rise, angle varies from 72 to 74 degrees.

(2) Thickness of from 1,000 to 1,100 feet, angle of seam from 27 to 30 degrees.

(a) towards the dip, angle varies from 63 to 60 degrees.

(b) towards the rise, angle varies from 78 to 77 degrees.

In 1911 Goldreich delivered a lecture before the Austrian Society of Engineers and Architects of Vienna on the "Theory of the Railway Subsidence in the Mining District, With Special Consideration of the Ostrau-Karwin Coal District." Since that date he has made several important contributions to the literature of this subject.

In 1912 Franz Bartonie discussed the causes of subsidence, but made no contribution to the theory.‡

In 1913 Goldreich published his work on "The Theory of Land Subsidence in Coal Regions with special Regard to the Railway Subsidence of the Ostrau-Karwin Coal District," and followed this with a volume entitled, "Land Movements in the Coal District and their Influence on the Surface."§

Goldreich criticises Jicinsky's contribution and theory in a number of points. He takes no exception to the fundamental principles but objects to the formula. Goldreich questions the assumption of a coefficient of increase of volume that will be applicable to all cases.

*Ballig, C. "Die Schätzung von Bergbauen nebst einer Skizze über die Einwirkung des Verbruches unter-irdischer, durch den Bergbau geschaffener Hohlräume, auf die Erdoberfläche," A. Becker, 1906.

†Padour, Anton Chapter on "Damage to the Land and Buildings" in the "Guide Through the Northwest Bohemian Brown Coal District," 1908.

‡Bartonie, Franz "Die Ursachen von Oberächenbewegungen im Ostrau-Karwiner Berg-Revier. Montanist, Rundschau, Feb. 16, March 1 and March 16, 1912.

§In press.

He points out the fact that the so-called "after-slide" of surficial material is not considered in the formula. He accepts Jicinsky's determination of the fracture in the coal measures, but objects to the formulating of general rules for determining the size of pillars. The principal portion of Goldreich's work is given to a discussion of the phenomena of subsidence in connection with railways. "From the profiles of sunken railway sections of the Ostrau-Karwin coal district it can be seen that these profiles have a parabolic form, that the maximum subsidences are found in the middle of these depressions, and that the amounts of subsidence decrease almost regularly towards the two ends of the curves until they finally become equal to zero." This regularity of the depressions caused Goldreich to undertake to formulate a theory of subsidence applicable to conditions such as those which exist in the Ostrau-Karwin district, the most distinctive feature being the surficial bed of plastic marl as much as 1,200 feet thick in places. Where the coal measures outcrop, the regularity of the surface depressions disappears and Goldreich takes refuge in the statement that we must depend merely upon experience.

Goldreich's observations developed the fact that following the subsidence of bed rock there is a vertical subsidence of the marl directly overlying and a lateral after-sliding of the adjacent and outlying marl. In discussing the subsidence of the roof strata he emphasizes the effect of the elasticity of each stratum. "When the elasticity of the subsiding roof strata is so great that the latter reach the floor of the worked out room without any disturbance in the coherence of the superimposed strata, then the volume of these subsiding strata remains unchanged." The subsidence of roof strata without increase of volume will occur in the case of the extraction of thin and flat seams. "The increase of volume which takes place during the first stage of the subsidence process is not enduring; for in consequence of the weight of the following roof strata the broken rock is again compressed, so that at the end of the rock movement there results a decrease of volume which is certainly not identical with the initial increase of volume." Only by observing the amount of surface subsidence caused by an underground working can a basis for estimating the coefficient of increase of volume under actual conditions be obtained. When the overlying beds are elastic there will be little increase in volume as the movement proceeds upward; under such conditions the term "harmless depth" cannot properly be used. "It cannot be pointed out strongly enough how absurd is the establishment of a harmless depth which should be valid for all work-

ings; the harmless depth has rather a theoretical character because the presuppositions required for the actual existence of the harmless depth are very seldom true in practice."

British Theories.

As previously noted, subsidences resulting from the mining of salt and coal in the British Isles were observed at an early date and were the cause of investigation by British engineers, who in general have supported the important principles of Belgian-French theories, although certain persons have taken exception to particular points in these theories. Numerous observations have been made upon subsiding areas and considerable valuable information has been collected, the data have been correlated and arranged, and empirical formulæ have been constructed so that adequate pillars may be left for the protection of surface structures and property of various kinds.

In 1868 a commission of Prussian engineers investigated subsidence in the various coal mining districts of England, and found that in England the opinion was approximately as follows:

(1) The working of coal at every known depth may affect the surface, but at depths greater than 400 meters (1,300 feet) it can cause damage only to certain buildings, such as cotton mills.

(2) In the case of complete extraction, filling may be a means of effective protection against movements of the earth.

(3) The practice of leaving pillars constitutes an efficient protection against the effects of exploitation upon the surface. The extent of these pillars of safety should be determined by the surface to be protected, the depth being known and the angle of rupture being assumed.

Observations carried on by J. S. Dixon demonstrated that the wave of maximum subsidence regularly followed the advancing face and that a wave of disturbance was just as regularly projected in advance of it;* that is, the wave of disturbance preceded the working face, but the maximum subsidence followed it. Joseph Dickinson called attention to the similarity between earth movements due to natural causes and those resulting from mining operations. He considered that "the direction of subsidence may be judged by analogy from the slopes taken by faults and mineral veins. The slope of a fault in horizontal strata averages about 1 in 3.07 from the perpendicular, varying according to the hardness and cohesion of the strata from

*Dixon, J. S. "Some Notes on Subsidence and Draw." Trans. Min. Inst., Scotland, Vol. 7, p. 224, 1885.

about 1 in 5 in hard rock to 1 in 3.75 in medium, and 1 in 2.5 in soft. He considered that for horizontal seams not exceeding 6 feet in thickness, and with strata of the average hardness of those in Lancashire, ordinary subsidence may be taken as extending on all sides to one-tenth of the depth, and that to obtain security a margin should be added. This margin is limited by some engineers to an additional one-tenth of the depth, while others add an arbitrary amount. When the strata are softer the extent of the subsidence is sometimes taken as one-sixth, or even one-fourth of the depth of the working, while, on the other hand, for hard siliceous rock, such as is found in South Wales, reductions are needed. He also agrees with other writers, that in seams of moderate inclination larger areas are required for support on the rise side than on the dip.”* T. A. O'Donahue in discussing subsidence† endorses the observations of J. Dickinson in the following language: “Joseph Dickinson is practically the only writer who has succeeded in connecting the threads of what was apparently a mass of contradictory evidence and in showing that the majority of cases approximately agree with a more or less definite rule.” In O'Donahue's opinion, which is the result of considerable experience in studying the effect of surface subsidence, including the taking of levels, the “breaking lines of strata may be estimated within narrow limits with average conditions.” He enumerates the important factors affecting the position of the breaking lines and the ultimate extent of the subsidence as (1) the relative hardness of the strata, (2) the inclination, and (3) the thickness of the coal seam. He also mentions the influence of surface deposits. He considers the various angles of draw that have been noted and points out that for safety the maximum angle for given conditions must be taken as the limit for safety. For coal beds 6 feet thick and overlying strata of moderate hardness, he has found that the angle of draw is from 5 to 8 degrees beyond the vertical. This means that if a pillar is to be left to protect objects on the surface, a margin of one-twentieth to one-tenth of the depth should be left in order to provide against the draw. With inclined strata the draw increases roughly in proportion to the degree of inclination of the strata. He accepts the normal theory as correct when applied to dips of 18 to 24 degrees, but only for dip workings. When the mine workings are on the *rise* the maximum draw is estimated at 8 degrees for strata

*Hughes, H. W. “A Textbook of Coal Mining,” 5th Edition, p. 185, London, 1904.
Dickinson, J. “Subsidence Due to Colliery Workings,” Proc. Man. Geol. Soc., Vol. 25, p. 600, 1898, and Colliery Guardian, Oct. 28 and Nov. 11, 1898.

†O'Donahue, T. A. “Mining Formulæ,” p. 244, Wegan, 1907.

nearly horizontal and the draw is taken to cease with strata at an inclination of 24 degrees.

The ideas of O'Donahue are expressed in his formula for shaft pillars as follows:

M = Margin of safety, say from 5 to 10 per cent of the depth,

D = depth of shaft,

X = distance at the seam, between two lines drawn from a point at the surface, one line being vertical and the other at right angles to the seam.

Shaft pillar in horizontal strata

$$\text{Radius of pillar} = M + \frac{D}{7}$$

In inclined strata

$$\text{Rise side} = M + \frac{D}{7} + \frac{2}{3} X$$

$$\text{Dip side} = M + \frac{D}{7} - \frac{1}{3} X$$

For seams less than 6 feet thick the size of the pillar may be decreased, while for thick seams it is suggested that the size of pillars determined for a 6-foot seam be multiplied by the "square root of the thickness of the seam in fathoms."

In discussing the effect of the thickness of the seam upon the amount of subsidence, O'Donahue calls attention to the effect of the material stowed in the goaf or gob. He makes the point that, other conditions being the same, the mining of a 6-foot seam would result in more than twice the vertical subsidence caused by mining a 3-foot seam, owing to the fact that little material is thrown into the gob in mining the 6-foot seam, while in mining the 3-foot seam undoubtedly much "brushing" would have to be done and, therefore, there would be considerable material left in the goaf or gob. Therefore, the total subsidence per foot of coal removed will be greater in the case of the thicker seam.

He objects to the statement that mining at depths of 1,800 to 2,000 feet will not cause subsidence, because careful levellings will show that the complete removal of the coal at even greater depths will cause a sinking of the surface. "When the whole of the mine is taken out subsidence of the surface follows at all workable depths. The writer's observations show that the working of a seam, for instance 4 feet thick, will cause the surface to subside about 3 feet if the seam be not greater

than 600 feet in depth, and will cause a subsidence of from 12 to 18 inches at a depth of 2,400 feet.

H. W. G. Halbaum has made a careful study of roof pressures in longwall work and has made some notable contributions to the theories of subsidence. In his study of the action of the roof in longwall mining* he called attention to the *locking of the roof* due to the lateral thrust in great roof sections. "The great roof sections, by successive slips, have descended a few inches. The motion has been arrested for the time being by the lateral thrust, and the great body of strata remains securely gripped in the powerful jaws of its natural clamps." Subsequently Halbaum formulated the following proposition: "Contained in the total force of the roof action, there is a horizontal component, the action of which is contrary to the direction of working, and the power of which is sufficient to deflect the roof action from the vertical line."†

After discussing carefully the planes of strain, Halbaum considers the cantilever idea. He likens the unsupported roof strata to a huge cantilever whose load consists primarily of its own weight. "It is evident that if the cantilever were homogeneous and if the neutral surface were at half-depth, and if efficiencies of the compressive and tensile stresses to propagate their respective strains were equal, we should, under such conditions, obtain a mean line which would be vertical; for the tensile and compressive components would be equal in length and equal, though opposite in obliquity. The obliquity of one component would thus exactly balance the opposite obliquity of the other, and the mean line would be vertical." He then points out that such a balancing of components is unlikely, for the resistance of ordinary coal-measure strata to compression is usually greater than their resistance to tension. The neutral surface of the cantilever must generally lie below the half-depth of the beam. Moreover, the beam is not homogeneous. "Viewed from the broader standpoint of internal nature and external environment combined, there must be little or no exaggeration in the statement that our cantilever is immeasurably stronger to resist compression than to resist tension; and hence we are bound to infer that its neutral surface is very low indeed and probably not many feet above the roof-line itself." "It follows that by far the greater portion of the absolute line of elementary strain is supplied by the tensile

*Halbaum, H. W. G. "The Action, Influence and Control of the Roof in Longwall Workings." Trans. Inst. Min. Engrs., Vol. 27, p. 211, 1903.

†Halbaum, H. W. G. "The Great Planes of Strain in the Absolute Roof of Mines." Trans. Inst. Min Engrs., Vol. 30, p. 175, 1905.

component, that by far the greater portion is projected over and towards the solid, and that the mean elementary line must, therefore, possess a normal obliquity little less in magnitude than that of the tensile component itself." Stated in brief the idea is this: "We start with a thick unloaded cantilever and we end with a thinner but loaded beam; thinner, because from the standpoint of their efficiency, the upper layers are gone; and loaded, because from the standpoint of their dead weight, the upper layers remain only as a true load on the effective beam beneath. This simultaneous thinning and loading of the effective cantilever seems probable for several reasons: The principal one perhaps is to be found in the fact that the original beam is a composite beam formed by an aggregation of smaller beams (strata) in superposition. The whole of the composite beam is an effective beam only so long as its several layers firmly adhere at their conterminous horizontal planes or boundaries. As soon as the uppermost layer (or series of layers) separates from its subjacent layer, or tends to slide thereon, it ceases at once to form any part of the effective cantilever, to which cantilever it must thenceforward sustain the relation of a load only." He calls attention to the fact that "when we examine the cases of natural subsidences of the earth's crust, we find that the great planes of strain, in the normal case, are always projected over and towards the solid (or unsubsidized) strata."

In a paper before the International Geological Congress, Professor George Knox summarized the various points "which may be considered sufficiently well established to form a basis for further investigations—namely:

(a) That surface subsidence invariably extends over a greater area than that excavated.

(b) The angle of pull is determined by the ratio between the excavated and subsided areas.

(c) That this ratio is determined by a large number of factors, among which may be included the following:

1. The amount of permanent support left in the unmined area.
2. The thickness of the seam worked.
3. The depth of the workings from the surface.
4. The method of working adopted.
5. The direction of working in relation to the jointing of the strata.
6. The rate at which the workings advance.

7. The nature of the strata overlying the workings.
8. The presence of faults, fissures, etc., in the strata.
9. The permeability of the overlying rocks.
10. The dip of the strata.
11. The surface contour.
12. The potential compressive forces existing in the strata containing the workings.”*

He concludes that the ratio between subsidence and draw must be the joint result of the forces liberated by the withdrawal of support from underneath the strata in the mined area. The larger the proportion of settlement resulting in subsidence the less can occur in the form of draw, and vice versa “The number of factors that may influence the results produced by the settlement of undermined strata is so great that only a wide and comprehensive inquiry by geologists and mining engineers in those countries where mining is conducted on a large scale can be hoped to provide sufficient evidence to establish a definite theory or theories to assist in overcoming some of the more common dangers due to subsidence.”

Alexander Richardson, in a paper before the Chemical, Metallurgical, and Mining Society of South Africa, took up the question of stresses in deep masses of rock unsupported for hundreds of feet horizontally. “Where the strata are unfaulted, one would be justified in considering the mass as a huge slab supported on two or more sides or as a lever hinged at the bottom of the workings. Over extensive areas the pressure on the roof of an excavation, assuming the bed to be horizontal, will become in time equal to the weight of the superincumbent strata; under no circumstances is it immediately so, since the overlying beds must have some carrying strength.”†

Opinions of American Engineers.

While no new theories have been advanced by American engineers it may be profitable to review their opinions as given to the public through papers, investigations, or testimony.

As previously noted, a number of prominent engineers have made investigations as to the nature, extent and cause of the damage to property resulting from surface subsidence in Scranton, Pennsylvania. The published and the special reports noted on page 28 include ex-

*Knox, George “Mining Subsidence.” *Proc. Int. Geol. Congress*, Vol. 12, p. 798, 1912.

†Richardson, Alexander “Subsidence in Underground Mines,” *Jour. Chem. Met. and Min. Soc. of S. Africa*, Vol. 7, p. 279, March, 1907; *Eng. and Min. Jour.*, Vol. 84, p. 196, 1907.

pressions of opinion, but little discussion of the principles and theories of subsidence.

Douglas Bunting, who has made a study of the "Limits of Mining under Heavy Wash" in the anthracite region of Pennsylvania,* considered the various sedimentary rocks of the coal measures and determined the minimum thickness of rock cover for various depths below the surface and for rooms of various widths. He had previously made a study of chamber pillars in deep anthracite mines, and had calculated the width of rooms for various depths upon the basis of the compressive strength of anthracite.†

In discussing subsidence in the longwall district of Illinois, G. S. Rice said, "The roof settles most in the first few months, but it is several years before it is entirely settled, by which time the gob has been squeezed down to one-half or one-third its original thickness." The roof is very free from slips and vertical cracks or joints until the coal has been mined below it, but when the coal is brought down in a long strip, it marks the roof just where the break of coal has occurred, and along these marks the roof afterwards breaks. These breaks seem to run up indefinitely, and oftentimes they can be followed up to the black slate, 8 or 10 feet above. As a result of mining the seam, which varies in thickness from 2 feet, 10 inches to 4 feet, or an average of 3 to 3½ feet, "the settling of the roof is appreciable at the surface even when the seam is at a depth of 400 or 500 feet; but so gradual is it and without vibration that the deep mines have caused no trouble in going under railroad tracks, and even under brick buildings, as has been done at La Salle."‡

Charles Connor believes "If we extract all the coal we, naturally, will have a subsidence of the surface. That must inevitably follow because, when the support is all removed, the rock settles down on the floor of the mine." He cited observations made in the county of Lanark, Scotland, where the mining of seven seams approximating 30 feet in thickness and lying at depths of from 900 to 2,700 feet necessitated the raising of canal banks 18 feet. The sinking was gradual and no water was lost out of the canal.§

In discussing the action of beds overlying mine workings

*Amer. Inst. Min. Engrs., Bul. No. 97, p. 1, 1915.

†Bunting, Douglas "Chamber Pillars in Deep Anthracite Mines." Trans. Amer. Inst. Min. Engrs., vol. 42, p. 236, 1911.

‡Rice, G. S. "System of Longwall Used in Northern Illinois Coal Mines," Columbia Univ. School of Mines Quart., Vol. 16, p. 344, 1894.

§Connor, Charles "Discussion of Paper." Proc. Coal Min. Inst. of America, p. 149, 1912.

W. A. Silliman expressed the opinion that these beds do not act as a monolith, but that the beds may have little adherence to each other. Where the measures are weakest the strain will be greatest, as in the fireclay bottom.*

S. A. Taylor has pointed out the fact that the tendency of one measure to slip on another is "counteracted by the fact that the roof is a continuous mass, so that as soon as any part tends to give, it has to slide to get past adjacent surfaces and the surfaces are pressed together with no little weight. If any part is weak, it is nevertheless held in place and the strain goes to the other measure." In his opinion the roof rock acts as a monolith in most cases.† He also believes that the occurrence or absence of subsidence depends on the height and character of the overlying strata. "You cannot set any hard and fast rule; the rule set down by the English authority 'one to ten' (there will be no breakage on the surface if the rock cover is ten times the thickness of the coal worked) will not hold. It may be true in some cases, but it will not serve as a universal rule, as its truth or falsity in any instance depends on the character of the overlying strata."‡

R. D. Hall has suggested that when the roof sags down over the edge of a pillar the curve of the roof tends to follow back over the solid coal, criticising the general notion that the roof lies flat upon pillar and then sags down over the edge of the pillar.§ He has also shown to what extent in his opinion shear is the cause of the failure of mine roof.¶ He concludes his discussion of shear, "In the case of mine roof, everyone seems to be confident that we have a structure which invariably fails from shear. The idea is contrary to all the evidence and should be dismissed. The raggedness of roof fractures disproves it if other reasoning does not."** In discussing the strength of mine roofs R. D. Hall has presented a series of sketches, showing conditions producing breakage of roof.†† He suggests that the roof over rooms acts after the first fractures not like a beam but like an arch, and that continuous beams or plates are replaced by disconnected arches or vaults. He concludes by suggesting a "progressive advance in demolition: First, a condition, as yet unnamed, symbolized by the tunnel in solid rock in which roof and sides and floor

*Proc. Coal Mining Inst. of America, p. 84, 1911.

†Op. cit., p. 85.

‡Taylor, S. A. In discussion of R. D. Hall's paper, "Effect of Shear on Roof Action," Proc. Coal Mining Inst. of America, p. 146, 1912.

§Hall, R. D. "Action of the Roof," Proc. Coal Min. Inst. of America, p. 64, 1911.

¶Hall, R. D. "Data of Petrodynamics," Mines and Minerals, Vol. 31, p. 210, 1910.

**Hall, R. D. "Effect of Shear on Roof Action," Proc. Coal Min. Inst. of America, p. 144, 1912.

††Hall, R. D. "The Strength of Mine Roofs," Mines and Minerals, Vol. 30, p. 474, 1910.

all partake of the beam strain. Second, a horizontal shear which converts the sides into mere supports and the roof into a true beam or plate; Third, a rupture of the roof which converts it into an arch, and finally, a failure of the arch or vault by one of the many weaknesses to which such structures are subject.”*

The tendency of the roof to arch has long been noted, and the mechanics of natural rock arches has been discussed by a number of engineers. However, there has been little agreement among engineers as to the portion of the burden of the overlying beds which is actually borne by such natural arches. The strata acting as a uniformly loaded

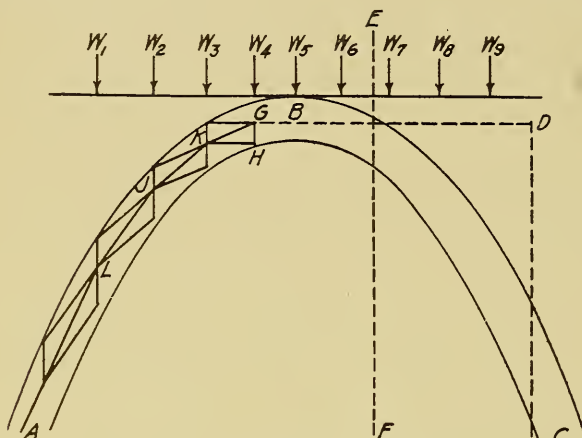


FIG. 40. STRESSES IN ARCH.

horizontal beam cannot support a great load, and as the strata sink the upper measures tend to arch and eventually the entire mass may be supported by the arch.

The theory of the arch as applied to this problem has been discussed by B. S. Randolph† as follows: In the arch $A B C$, Fig. 40, the two sides $A B$ and $B C$ are mutually supported at B where the thrust is horizontal. Assuming the load to be evenly distributed over the arch, it is found that the points B, G, K, J, L all lie in the line of stress. This line of stress “when lying in solid material over an excavated cavity will constitute, for all practical purposes, an arch supporting all the material above it and allowing the removal of all the material below it up to the point where this material becomes effective

*Hall, R. D. “The Last Stand of the Mine Roof.” Coal Min. Inst. of Amer., 1914, and Coal Age, Vol. 6, p. 982, 1914.

†Randolph, B. S. “Theory of the Arch Applied to Mining.” Coll. Engineer, Vol. 35, p. 427, 1915.

in resisting the stress. There will, of course, exist along and on each side this line of stress a zone of material under more or less pressure, depending for its width on the total stress and the elasticity of the material. The position and character of the forces acting on the arch will vitally affect the shape of this line of stress. In an arch under a perfect fluid, where the pressures are all radial acting toward a common center, the line of stress becomes the arc of a circle. With an excess of load toward the center, it takes the shape of the parabola, the focal distance shortening as the central load exceeds that on the side. With the excess of pressure on the sides, say at an angle of 45 degrees, it assumes the shape of an ellipse, the focal distance shortening as the pressures at the side exceed those in the middle." In the arch formed over rooms, as the load for all practical purposes is equally distributed, "the curve will be a parabola with a longer or shorter focal distance

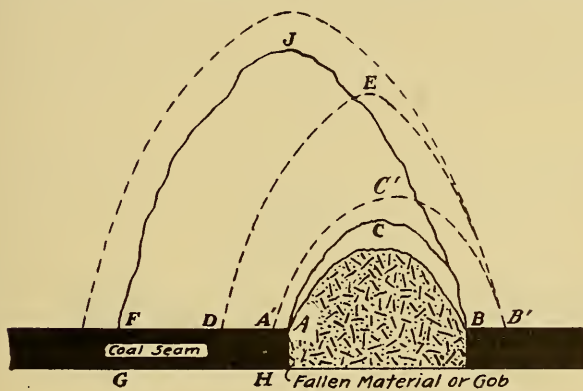


FIG. 41. ARCH STRESSES IN MINE ROOF.

depending on the nature of the strata." "Let Fig. 41 represent the section of a coal seam from which the coal has been removed between A and B, the roof having fallen to the irregular line A C B. The dotted line A' C' B' will indicate the line of stress. This, it will be seen, impinges on the coal close to the edge at A. The stress at this point represents half the weight of the strata overlying the span A B which is assumed to be sufficient to crush the coal about the point A. The integrity of the arch being destroyed, the line of stress must seek a new position such as D E B. Naturally this movement will be no greater than is absolutely necessary to gain a solid footing for the arch, which will again be so near the edge of the coal already crushed that it will fail again in a short while, necessitating a further adjustment

of the position of the arch. With this continuing failure and readjustment we have the well-known phenomena of a crush or squeeze advancing slowly over the workings, destroying coal as it goes.

If now a considerable body of the seam, as $A H G F$, is quickly removed a "fall" may result which will reach high above the seam, say to the line $F J B$, which will cause the line of stress to move quickly and reach the coal well back from the point F , where it is sufficiently solid to give the needed support, and the working will be said to have "gotten ahead of the crush," when in fact the crushing force has gone ahead of the working. This explains the common experience of the relief at the working end of the pillar caused by an extended break in the roof over the exhausted area.

Under other conditions, especially when the arching line of stress has a wide span, thus carrying a large amount of weight, the crushing force may prove too much for even the solid coal well back from the end of the pillar and cause the phenomena of crushed coal, broken tim-

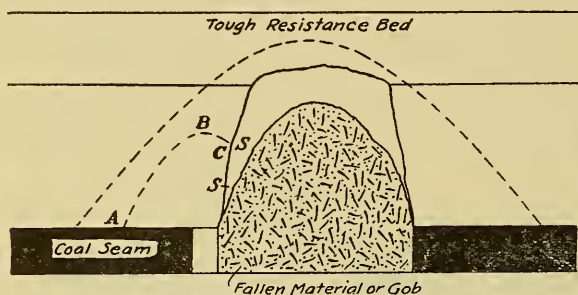


FIG. 42. SPACE SHORTENED BY FALLING OF ROOF.

bers, creeping floor, etc., well down the room or stall, while the ends of the pillars will be free from any trouble, as they carry only the small amount of material which is below the line of stress. This condition will sometimes be cured automatically by the material falling from the top of the cavity over the exhausted area in such a manner that the space between the material already down and the undisturbed measures will be filled and the opposite limb of the arch (the right hand in the figure) will find support on this already fallen material and thus shorten the span of the arch and lessen the total weight, as illustrated in Fig. 42.

When the break has reached the surface, this filling takes place more rapidly owing to the fracture of the overhanging beds along the edge of the break and, since the arch has a new point of support for

its inner or right hand limb, the conditions are ripe for further working undisturbed under the smaller arch.

While the general shape of the line of stress in the cases under consideration is the parabola, for all practical purposes the ratio between the span and the rise or height of the arch will vary much as the material varies in which it exists. After the fall of the first mass the cavity grows through the crushing and falling of the material along the top of the arch, due to the pressure along the line of stress, and by the splitting off along the joint planes of the material on the sides due to the same cause. Since the pressure along the upper portion of the line of stress is manifestly less in a high sharp arch than in a low flat one, the shape of the arch in this respect may be expected to vary with the capacity of the material to withstand this stress. Hence, there will be a high arch in a soft material with numerous joints and a flat arch in tough material with fewer joints. This may be verified practically by the examination of old drifts or tunnels where the overlying material has had an opportunity to fall and take the shape due to such conditions without regard to other influences.

If, then, the cavity in its upward progress encounters a bed of tough resistant shale or sandstone, it may fall so slowly that a large area may be opened by continuous mining during the delay, resulting in a heavy weight along the line of stress due to the wide span and crushing the coal either at the working end of the pillar or at such point along the course of the room as the line of stress may meet the coal as shown in Fig. 42. Such a crush is not likely to find relief until the overlying measures are sufficiently broken down to fill the space $S S$, and allow the development of a new smaller arch of stress $A B C$, which, having less span and consequently less load, will transmit less load to the point A .

Dr. F. W. McNair has reviewed the question of pressures and support in the deep copper mines of the Lake Superior region. In a lode dipping 38 degrees and with pillars 50 feet wide, having on either side an open space of 150 feet, the pressure on the pillar at a depth of 5,000 feet would be 1,239 tons per square foot, allowing for neither rigidity nor arching and supposing the weight on the pillar evenly distributed. The pillar would fail under this pressure if it were mainly trap rock. "As a matter of fact, in such a case, the rigidity of the mass distributes a large part of the load out over the rock beyond the walls of the opening. That this rigidity may be considerable is illustrated in several cases in which areas of hanging wall as wide as 200 feet or more have

no support between walls and yet have stood up for several years. As the rock between pillars and walls bends downward the tendency is to concentrate the load at the edge or face of the pillar or walls. The outer parts of the pillar may thus become overloaded and fail by the splitting off of pieces of rock, that break from the base as well as the top, and like any hard rock under a crushing load, the pillar usually fails suddenly. The hanging rock mass moves, of course, when the pillar crushes, and the vibration due to the sudden though slight displacement is often conveyed to the surface. The result is a miniature but perfectly genuine earthquake that may be felt over a distance several times that of the pillar from the surface. With the crushing of the pillar and the movement of the hanging wall, a readjustment of the weight takes place, and the process begins over again. Eventually, at great depths, the hanging and foot walls must come together.

"The readjustments that take place when a pillar fails sometimes put an enormous longitudinal thrust on the foot wall, and in places its surface portion has buckled under such stress. Experience seems to show that at the great depths recently reached it is useless to expect to hold up the hanging rock mass for a long time by any scheme of pillars unless far too much of the lode is left in place, and that the only feasible method is to cut away the entire lode and permit the hanging to cave as rapidly as it will to the point where the broken rock fills again the whole space and redistributes the weight over the foot wall."*

C. T. Rice objects to the general statement that stopes will cave until filled, except in the case of running ground. In the few caved stopes which he has inspected he has "always found an open space between the arched roof and the pile of caved rock. In general, such a large stope opening is necessary before caving commences; the self-supporting dome is assumed before the stope fills itself. The caving action is progressive, and as the slabs accumulate in the stope they so support the sides that caving ceases. Finally, owing to the weakening of other stopes, the faulting stage is reached; not until then does the opening become completely filled.

"In supporting the roof of a stope, only that portion of the roof that is below the line of the dome of equilibrium requires support; the rock above this dome sustains itself. If, therefore, the shape of this dome of equilibrium in each kind of rock were known, it would be easy

*McNair, F. W. "Deep Mining in the Lake Superior Region," *Min. and Sci. Press*, Vol. 94, p. 275, 1907, and *Eng. and Min. Jour.*, Vol. 84, p. 322, 1907.

to calculate the weight of rock hanging below the dome, and so timber the stope as to hold up this weight." C. T. Rice is under the impression that the shape of this dome is fairly constant in each kind of rock; especially in the same rock in the same district. "Of course, slips and joints, sudden changes in chemical composition, the dip of the strata in sediments, and many other facts, would affect the shape of the dome, but as long as these were small their effect would also be small. If investigation of the shape of this dome should suggest any formula to determine the strength of timber necessary to support the ground below the dome, the effect of these joints, etc., could easily be included by the factor of safety used."*

*Rice, Claude T. "The Dome of Equilibrium and the Caving System of Mining." Mining and Sci. Press, Vol. 95, p. 85, 1907.

CHAPTER IV.

ENGINEERING DATA AND OBSERVATIONS.

In America very few data have been collected on subsidence due to mining operations, at least the data, if collected, have not been made available for scientific purposes.

In order that observations may be of value the following correlated data are desirable:

(1) The elevations of a number of points on the surface for a period of years both prior to, during, and following the mining directly beneath.

(2) The position of these points with regard to permanent stations located outside of the mining field or upon ground which has not been or will not be subject to the influence of the mining operations.

(3) The position of the working face in the mine on the various dates of survey.

(4) An accurate location and description of the character of the portions of the mineral deposit left unmined.

(5) An accurate location and a description of the supporting materials placed in the excavated area.

(6) The thickness and dip of the material mined.

(7) The thickness and character of the bed immediately underlying.

(8) The thickness, dip, and character of the overlying rocks and all available information in regard to structure.

(9) The thickness and character of the surficial material.

(10) The quantity of water removed from the mine.

(11) The location, extent, and data of underground movements of rocks overlying the mineral deposit.

In Europe records have been kept for many years in various districts in order to determine the vertical amount, lateral extent, rate, and duration of subsidence.

Among the first surveys made to determine the movement of the surface were those of Fayol.* At Commentry Mine from 1879 to 1885, as shown in Figs. 43 and 44, surveys were made to correlate surface movement and the advance of the working face. The seam which was almost 48 feet thick was worked in horizontal slices of about 8 feet in

*Fayol, H. Bul. Soc. Ind. Min., II ser., Vol. 14, p. 818, 1885. Coll. Eng., Vol. 11, p. 26, 1890-91.

ascending order. The thickness of rock cover was 321 feet. Some filling of shale and sandstone quarried on the surface was used. It was observed that:

(1) During the removal of the first slice, the lowering of the surface gradually grew greater, and was further increased considerably by the working of the second.

(2) The area of subsidence was about four times larger than the area worked.

(3) The maximum sinking was 3 feet 5 inches or one-fifth of the height of the two slices.

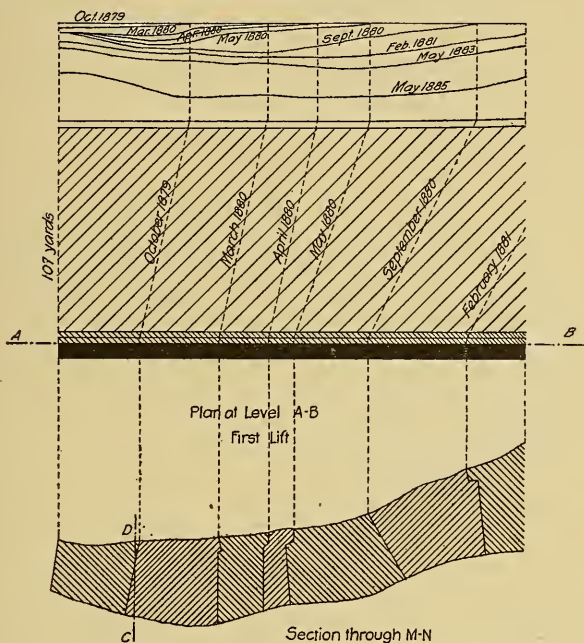


FIG. 43. SUBSIDENCE AT COMMENTRY MINE.

(4) The movements of the ground appeared at first at a certain horizontal distance in advance of the working faces and this distance remained nearly constant.

(5) The subsidences increased during a certain time while the working proceeded.

(6) The second lift caused a total subsidence almost equal to that of the first lift. This subsidence was 2 feet 1 inch for the first and 1 foot 11 inches for the second, in all 4 feet.

(7) The area of subsidence cannot be determined, either by normals or verticals from the bed worked.

The surveyor's records showing surface movement in connection with the Warrior Run Mine disaster have been noted previously.* The ratio between the volume of subsidence as noted on the surface and the volume of excavation has been noted for the caving system of mining on the Gogebic range.† "When a slope caves, and the dome above it runs up into sand or loose rock, the depression formed is usually in the shape of an inverted cone; but where the ore body is wide, or deep below the surface, the subsidence usually takes the form of terraces. Sometimes comparatively large areas will break through cleanly and the whole surface will drop suddenly and as a unit, but this is exceptional.

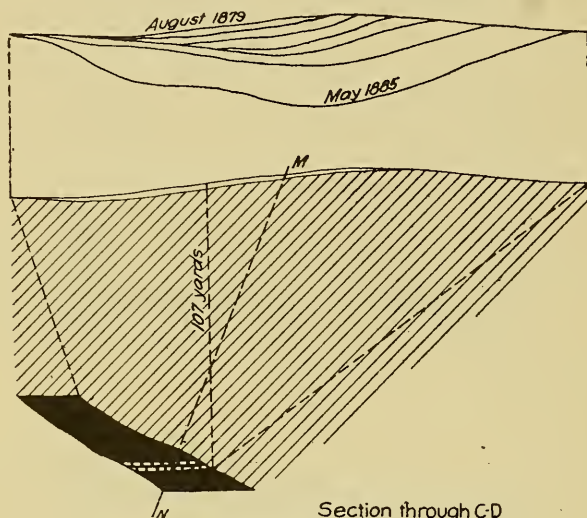


FIG. 44. SUBSIDENCE AT COMMENTRY MINE.

After the back has once started to cave, the surface usually sinks in terraces." In the area under observation the deposit consisted of a lense of soft hematite about 40 feet in average width and 150 feet high, with a length of nearly 1,600 feet on the incline, lying in a trough between a dike of diorite and a thick band of slate. The trough pitched 11 degrees, the hanging wall was hard jasper, and the ore was mined first by square-set rooms and pillars, about 60 per cent of the ore being secured on first mining, but later the pillars were robbed. The hanging wall has dropped from 15 to 75 feet and subsidence has extended to

*See page 43.

†Eaton, L. "Surface Effects of the Caving System," Min. and Sci. Press, Vol. 97, p. 428, 1908.

the surface. The proportion between the volume of surface subsidence and the volume of excavation is shown by the following figures:

	Cubic Feet
Original volume of ore body.....	7,680,000
Present volume of ore body, including rock, timber, etc.....	2,360,000
Volume of material removed.....	5,320,000
Volume of material removed under cave.....	2,300,000
Volume of material removed under subsidence on surface.....	3,020,000
Volume of cave on surface.....	790,000
Volume of subsidence on surface.....	1,430,000
Volume of cave on surface	1
	=
Volume of ore taken from under it	2.91
Volume of subsidence on surface	1
	=
Volume of ore taken from under it	2.11

J. S. Dixon made observations at Bent Colliery* A line was selected at right angles to the advancing workings and as nearly as possible on the level course of the coal. Stations were put in every 100 feet and afterwards every 50 feet. The excavation was 5 feet 6 inches high and the overlying strata were allowed to fall and fill it. The surface was principally boulder clay. The original level of the surface before the pillars were drawn is shown in Table 8.

The pillars were removed for a distance of 240 feet back from the solid coal on Jan. 21, 1882, and no subsidence of the surface had ensued. On May 27, 1882, the levels showed the maximum subsidence to have been 1.80 feet at station 1650, which was 145 feet back from the face, and the draw extended to 60 feet in advance of the working face. On November 14, 1882, the face was 610 feet from the solid, and the subsidence was as is shown in the table. On April 15, 1883, the face was 750 feet from the solid; on November 27, 1863, it was 1,060 feet distant; and on October 23, 1884, the removal of the pillars had been completed for some months, and the face was 1,230 feet from the solid. On June 17, 1885, the workings had been in the same position for about a year. On December 4, 1885, it was found that subsidence had practically ceased and the draw had not altered.

The conclusion arrived at is that subsidence from the removal of coal in this case attains its maximum towards the center of the excavated space, and gradually decreases in each direction. The maximum subsidence, 4 feet, was 73 per cent of the thickness of the coal, and the average, 3.76 feet, was 68 per cent. The wave of maximum subsidence regularly followed the working face at an average distance back of 186

*Trans. Mining Inst., Vol. 7, p. 224, Scotland, 1886. Cit. in Brough, B. H., "A Treatise on Mine Surveying," pp. 241-245.

TABLE 8.
OBSERVATIONS AT BENT COLLIERY.

Peg.	Original Level of Surface	Subsidence From Original Level at					
	1881	Nov. 14, 1882	April 5, 1883	Nov. 27, 1883	Oct. 23, 1884	June 17, 1885	Dec. 4, 1885
600	640.6				0.25	0.35	0.45
650	648.9				0.35	0.60	0.60
700	657.2				0.77	0.94	0.94
750	664.6				1.18	1.27	1.40
800	667.5			0.23	1.37	1.75	2.00
850	673.1			0.63	1.50	2.24	2.34
900	675.6			1.13	2.57	2.74	2.82
950	676.0			1.61	2.97	3.14	3.22
1000	677.1			2.10	3.27	3.49	3.60
1050	677.3			2.43	3.32	3.64	3.75
1100	678.8		0.50	2.80	3.52	3.80	4.00
1150	679.7		0.70	2.93	3.57	3.81	3.92
1200	680.6		1.20		3.52	3.52	3.52
1250	680.9	0.60	1.60	3.08	3.45	3.45	3.45
1300	679.2	0.40	2.00	3.03	3.42	3.42	3.42
1350	677.9	1.60	2.25	3.00	3.27	3.27	3.27
1400	677.5	1.60	2.45	3.33	3.17	3.17	3.17
1450	680.8	2.30	2.90	3.42	3.42	3.42	3.42
1500	680.1		3.05	3.05	3.05	3.05	3.05
1550	677.1	2.90	3.20	3.20	3.20	3.20	3.20
1600	675.5	3.00	3.00	3.00	3.00	3.00	3.00
1750	672.8	2.80	2.80	2.80	2.80	2.80	2.80
1850	663.4	1.70	1.70	1.70	1.70	1.70	1.70
1950	648.6	0.60	0.60	0.60	0.60	0.60	0.60
2000	649.7	0.04	0.04	0.04	0.04	0.04	0.04

feet, or 1 foot horizontal for each $3\frac{1}{2}$ feet perpendicular. The permanent lengths of the draw may be taken as 100 feet on one side and 83 feet on the other. At these points the depth of the coal was 650 and 646 feet, representing a draw of 1 horizontal for each 7.14 feet perpendicular on the average. The coal dips at 1 in 20.

Surveys were made at the South Kirby Colliery in order to determine the extent and amount of surface movement due to the removal of a shaft pillar.* No observations were made until two years after the mining of the pillar was commenced. The seam in which the pillar was removed lies at a depth of 2,108 feet, is 3 feet 9 inches thick, and dips 1 in 18. Above the coal is one foot of clod. Some movement of the surface was noted before the surveys were made. At depths of 1,600 and 1,800 feet occur the Beamshaw seam of 3 feet and the Barnsley seam of 9 feet which had been worked previously. The data of the surveys are given in Table 9. The unusually large ratio of subsidence (3.47 feet) to the total thickness excavated in removing the pillar (4.75 feet) is attributed to the failure of the shaft pillar in the overlying Barnsley seam.

*Snow, Charles "Removal of a Shaft Pillar at South Kirby Colliery." Trans. I. M. E., Vol. 46, p. 8, 1913.

TABLE 9.

DATES OF LEVELING AND PARTICULARS OF SUBSIDENCE AT SOUTH KIRBY COLLIERY.

Dates of Levelings	Subsidence Since Previous Leveling	Total Subsidence
	Feet	Feet
November, 1903.....	Nil.	Nil.
July, 1904.....	Nil.	Nil.
March, 1905.....	Nil.	Nil.
October, 1905.....	0.74	0.74
April, 1906.....	0.17	0.91
October, 1906.....	0.22	1.13
June, 1907.....	0.77	1.90
April, 1908.....	0.17	2.07
December, 1908.....	Nil.	2.07
May, 1909.....	Nil.	2.07
May, 1910.....	0.82	2.89
May, 1911.....	0.18	3.07
June, 1912.....	0.40	3.47
June, 1913.....	Nil.	3.47

Levelings made by Chas. Snow at the Hickleton Main Colliery (details not given) showed that subsidence was evident 433 feet in advance of a rapidly advancing longwall face, and that total subsidence occurred

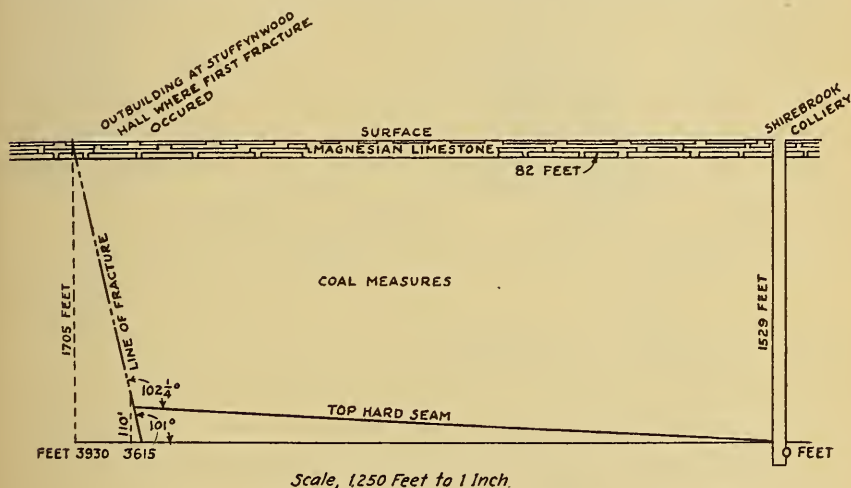
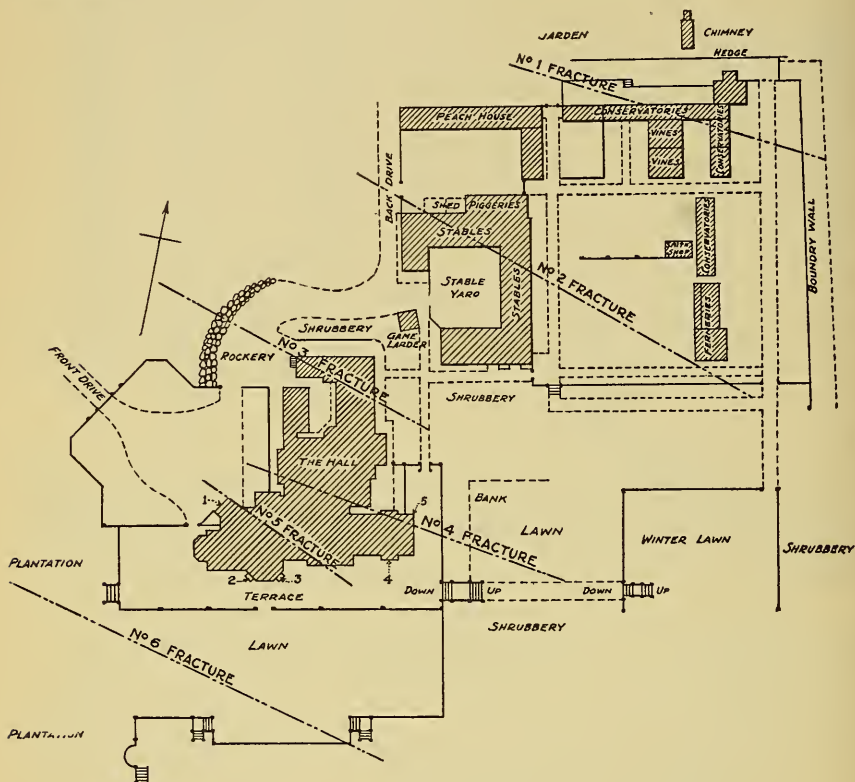


FIG. 45. ANGLE OF FRACTURE AT SHIREBROOK COLLIERY.

666 feet back from the face, the amount of subsidence being 4.5 feet. At the edge of the shaft pillar, 500 feet back from where the greatest subsidence occurred, the subsidence was 1.28 feet. Subsidence extended for a distance of approximately 600 feet over the shaft pillar.*

*Trans. I. M. E., Vol. 46, p. 21, 1913.

For a period of 16 years surveys were made at the Teveral and Pleasley Collieries by J. Piggford,* but details of the surveys have not been published. The angle of draw or fracture was estimated to be approximately 16 degrees from the vertical and toward the unworked coal. The depth of the coal seam was approximately 600 feet and the coal was from 5 to 6 feet thick.



Scale, 120 Feet to 1 Inch

FIG. 46. FRACTURES AND SURVEY STATIONS, SHIREBROOK COLLIERY.

Records were kept at Shirebrook Colliery and reported by W. Hay.† The coal lies at a depth of from 1,500 to 1,700 feet, dips 1 in 24, and is 5 feet thick. When the longwall face was 240 feet from Stuffynwood Hall cracks were noted in the surface, the direction of fracture varying as much as 15 degrees from the direction of the coal

*Trans. I. M. E., Vol. 38, p. 128, 1909.

†Hay, W. "Damage to Surface Buildings Caused by Underground Workings." Trans. I. M. E., Vol. 36, p. 427, 1908.

face. Fig. 45 shows the angle of fracture in section. When the working face was almost vertically beneath, the cracks had attained their maximum width and thereafter commenced to close. When the face had advanced 300 feet farther, the walls of the buildings had assumed practically their normal position.

Levels taken at regular intervals are given in Table 10. The survey stations are indicated on Fig. 46.

TABLE 10.
SUBSIDENCE AT STUFFYWOOD HALL.

Date	Time in Months From First Levels	Station No. 1a	Station No. 1	Station No. 2	Station No. 3	Station No. 4	Station No. 5
March 7, 1906	0	0.00	0.00	0.00	0.00	0.00	0.00
May 18, 1906.	2	0.00	0.00	0.00	0.00	0.00	0.00
June 11, 1906.	3	0.00	0.00	0.00	0.00	0.00	0.00
June 20, 1906.	3	0.00	0.00	0.00	0.00	0.00	0.00
July 10, 1906.	4	0.02	0.00	0.01	0.02	0.02	0.03
Aug. 27, 1906.	5½	0.11	0.09	0.16	0.18	0.18	0.20
Oct. 25, 1906.	7½	0.19	0.17	0.17	0.19	0.18	0.20
Dec. 10, 1906.	9	0.20	0.18	0.17	0.19	0.17	0.21
Feb. 6, 1907..	11	0.22	0.18	0.17	0.19	0.23	0.29
Mar. 9, 1907.	12	0.26	0.26	0.17	0.25	0.31	0.37
Mar. 29, 1907.	12½	0.28	0.26	0.26	0.25	0.31	0.37
June 20, 1907.	15½	0.67	0.43	0.44	0.44	0.52	0.63
Aug. 21, 1907.	17½	0.86	0.67	0.67	0.68	0.80	0.91
Nov. 12, 1907.	20	1.09	0.84	0.83	0.84	0.99	1.10
Jan. 30, 1908.	23	1.23	0.96	0.91	0.97	1.15	1.26
July 9, 1908..	28	1.56	1.23	1.21	1.34	1.41	1.51
Nov. 12, 1908.	32	1.74	1.34	1.35	1.37	1.54	1.63

The maximum subsidence was 1.74 feet and the minimum 1.34 feet, the average being practically 30 per cent of the total height of excavation.

Levels extending over a period of five years were taken by S. R. Kay on the surface of a portion of two collieries mining at depths of 360 feet and 990 feet.* Where the levels were run, the surface was fairly level, the strata were nearly horizontal and were free from faults of any magnitude. The strata consisted of alternating beds of shale, sandstone and limestone, none being massive. Figs. 47 and 48 show the data secured. The working of the 5-foot seam at a depth of 360 feet resulted in subsidence amounting to practically 70 per cent of the thickness excavated. Similar effects resulted from mining the 3-foot, 6-inch bed. At the greater depth subsidence began about six months after the coal had been mined and continued for years.

*Kay, S. R. "Effect of Subsidence Due to Coal Workings." Proc. I. C. E., Vol. 135, p. 115, 1898.

A report by C. Menzel* showed that since 1885 observations of the rate of settlement had been made at eight-two points in the vicinity of the collieries of Zwickau, Saxony. The depth of the coal beds varies from 600 to 2,400 feet. A maximum subsidence of 7.1 feet was noted twelve years after three seams had been mined out at a depth of from

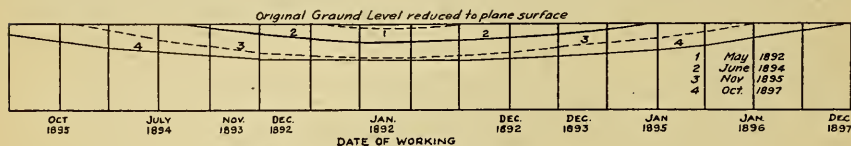


FIG. 47. DATA OBTAINED BY S. R. KAY.

600 to 900 feet. At a depth of 1,500 feet the subsidence was only 0.6 feet. By the use of filling subsidence was greatly reduced, it being noted that on an average the filling was compressed to one-half of its volume when stowed. The ratio of subsidence to thickness of seam excavated was found to vary from 1:1 to 1:7, the average being 1:2. Frenzel suggested this latter ratio for shallow seams.

Numerous observations have been made in Germany during the last thirty years. R. Hausse has reported upon the angle of break, angle of draw, and the coefficient of increase of volume. Jicinsky, Gold-

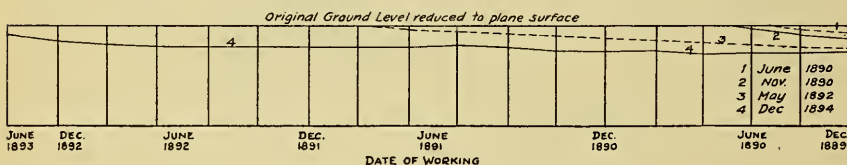


FIG. 48. DATA OBTAINED BY S. R. KAY.

reich, and others have reported upon subsidence in Austria-Hungary but in general these data have been secured in districts where the coal measures are covered with heavy beds of marl. From the foregoing statements of observations the following may be presented as representative in so far as general statements can be made to apply to mining operations each of which is conducted under different geological conditions.

ANGLE OF BREAK AND DRAW.

Dr. Niesz has made many observations upon subsidence, particularly on the angle of fracture in various kinds of rock and on the com-

*Menzel, C. "On the Relation of Surface Subsidence to the Thickness of Worked-Out Coal Seams at Zwickau." Abs. Proc. I. C. E., Vol. 140, p. 331. Jahrbuch für B.-, u. Hüttenwesen im K. Sachsen, p. 147, 1899.

pressibility of filling. He states that "the angle of fracture of limestones, conglomerates, etc., is found to be from 45 to 48 degrees—nearly the angle of repose. In quicksand the angle is greater, while in clay, slate, and marl it may be 60 degrees, and in stone under favorable conditions even 75 degrees. Sandstones with silicious binding material are ranked as nonplastic strata. Initial subsidences in these are followed by others, but at longer intervals than in plastic strata. The angle of fracture is generally not less than 82 degrees."*

Dr. J. S. Dixon reported, "In a level seam about 6 feet thick, by careful leveling on the surface prior to and after working, it was found that the draw or angle of subsidence of the strata was about 76 degrees from the horizontal plane."†

H. F. Bulman says that in a seam dipping 1 in 10, the lines of break extended over the solid coal forming an angle of 45 degrees with the horizontal on rise workings, 50 degrees in level workings, and 56 degrees on dip workings. In a wide goaf area the average inclination of the planes of fracture was 68 degrees from the horizontal plane; and at the rise side of a shaft pillar, the inclination was roughly 58 degrees from the horizontal plane over the solid coal.‡

S. R. Kay|| has presented the following formula for determining the radius of support:

$$r = \frac{3 \sqrt{d} x \sqrt[3]{t}}{0.8}$$

r = radius of support in feet,

d = depth in feet,

t = thickness excavated in feet.

This allows for the angle of break or draw.

Joseph Dickinson says, "the direction of subsidence may be judged of from the slopes of faults and mineral veins." He gives these slopes as 1 in 5 for hard rock, 1 in 3.75 for medium rock, and 1 in 2.5 for soft rock.§

O'Donahue says that the angle of break will be from 5 to 8 degrees beyond the vertical for horizontal beds, and that the maximum draw on dip workings will be 24 degrees; he finds the same angle to be the limit for workings to the rise.**

*Zeit. für Berg., Hütt., u. Salinenwesen, Vol. 58, p. 418, 1910.

†Trans. Inst. Min. Eng., Vol. 34, p. 416, 1907.

‡Trans. Inst. Min. Eng., Vol. 34, p. 417, 1907.

§Proc. Inst. Civ. Eng., Vol. 135, p. 149, 1898.

§Trans. Manchester Geol. Soc., Vol. 25, p. 600, 1885.

**O'Donahue, T. A. "Mining Formula," p. 248.

O'Donahue* offers two formulæ to determine the angle of draw:

$$d = 8 + \frac{2}{3} D$$

$$d' = 8 - \frac{1}{3} D, \text{ in which}$$

D = inclination of seam in degrees,

d = angle of draw toward dip workings,

d' = angle of draw toward rise workings.

E. H. Robertson gives a rule for shaft pillars (used in Northumberland and Durham) which allows for the angle of break and draw:

$$\text{Radius of shaft pillar in feet} = \frac{D}{6} + 2\sqrt{Dt},$$

D = depth of shaft in feet,

t = thickness of seam in feet.

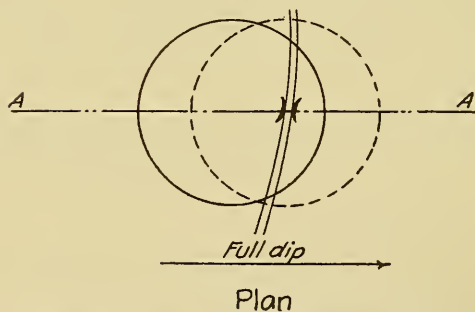
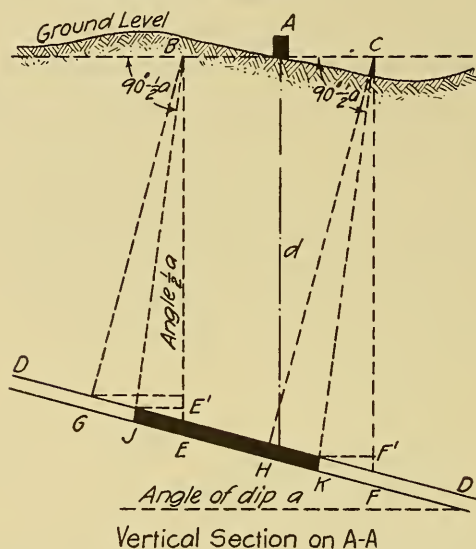


FIG. 49. LOCATION OF SHAFT PILLAR IN DIPPING BED. (O'DONAHUE.)

*O'Donahue, T. A. "Mining Formula," p. 248.

Hausse estimated that in general the line of fracture will be between the vertical and the normal to the seam. In addition to the line of main fracture, Hausse refers to the secondary break or draw. He says that in case of horizontal beds this line of secondary break is situated along the bisector of sliding materials of the supplementary angle of the natural slope.

The effect of the dip of the strata has been considered by many authors in their discussion of the simplest cases, in fact, most of the formulæ for angle of break consider the dip of the strata.

Gonot's law of the normal and Schulz's rule, the earliest of the theories, considered the angle of dip. As previously noted, Hausse, following Jicinsky, supports the theory that the angle of break will fall midway between the normal to the seam and the vertical. From a careful study of the subsidence occurring in the Saxon coal field R. Hausse determined the direction of the plane of fracture by the following formula:*

$$\begin{aligned} a &= \text{angle of fracture,} \\ d &= \text{dip of strata,} \\ \tan a &= \frac{1 + \cos^2 d}{\sin d \cos d}, \text{ in which,} \\ \text{if } d &= 0^\circ, \quad \tan a = \infty \quad \text{and } a = 90^\circ \\ \text{and if } d &= 90^\circ, \quad \tan a = \infty \quad \text{and } a = 90^\circ. \end{aligned}$$

S. R. Kay suggests that for inclined strata the angle of fracture will be midway between the perpendicular to the seam and the vertical. If the angle between the perpendicular to the seam and the vertical is a , then the pillar necessary to protect a given object on the surface must be shifted, on account of the dip, from a position directly beneath the object by an amount equal to $d \tan \frac{1}{2} a \cos a$, in which d equals the depth.

Goldreich gives Table 11 showing the angle of break according to the most important theories.†

*Results actually obtained in practice confirm this theory. Thus, for supporting the glass works at Doehlen, in Saxony, a 76.8-foot pillar was left; nevertheless the surface sank considerably. The coal seam dipped 12° and was 540 feet deep. Calculated from the depth and size of the pillar, the angle of fracture was found to be 82° , or $2^\circ 20'$ less than the result obtained from the theoretical formula. In another case in the same district the value of a was found to be $82^\circ 30'$, or $1^\circ 50'$ less than that found theoretically. (Brough, B. H. Proc. Inst. Civ Engrs., vol. 135, p. 150, 1898.)

†Goldreich, A. H. "Die Theorie der Bodensenkungen in Kohlengebieten," p. 42.

TABLE 11.
ANGLE OF BREAK.

Dip	Rule of Hausse		Rule of Thiriart	German Rule		Rule of Jicinsky	English Rule
	First Hypothesis	Second Hypothesis		First Hypothesis	Second Hypothesis		
0	70	70	Toward the Dip	70	70	90	81¼
10	65	67	70	75	70	85	73¾
20	60¼	64¼	67½	75	70	80	66¾
30	56	62	65¼	70	60	75	64
40	52¾	60¾	63	60	55	70	64
50	50¾	60¾	61¼	55	55	70	64
60	51	61	60¼	55	55	75	64
70	54	61	60½	55	55	80	64
80	61½	65½	62	55	55	85	64
90	70	70	65¼	55	55	90	64
			70	55			
0	70	70	Toward the Rise	70	70	90	81¼
10	75	77	70	75	79	85	73¾
20	79¾	83¾	72½	75	79	80	66¾
30	84	90	74¾	75	70	75	90
40	87¼	95¼	77	75	70	70	90
50	89¼	99¼	78¾	75	70	70	90
60	89	99	79¾	75	70	75	90
70	86	93	79½	75	70	80	90
80	78½	82¼	78	75	70	85	90
90	70	70	74¾	75	70	90	90
			70	75	70		

AMOUNT OF SUBSIDENCE.

Various writers have attempted to express the amount of subsidence as a percentage of the thickness of the seam worked. In Table 12 data from various districts are assembled, showing the depth of the workings, the thickness of the coal mined, and the vertical amount of subsidence expressed as a percentage of the thickness of the material removed.

TABLE 12.
AMOUNT OF SUBSIDENCE EXPRESSED IN PERCENTAGE.

Depth in Feet	Percentage Subsidence	Thickness of Coal Required Feet	Filling	Locality	Authorities
360	70.0	5.0		England	S. R. Kay
990	64.0	3.5		England	S. R. Kay
.....	46.0	29.36	Stowing	Bully-Greenay, France, England	O'Donahue
432	44.4	30.0			O'Donahue
600	75.0	4.0			Dixon
2400	25.0	4.0			Fayol
650	68.0	5.5		England	
745	19.0	7.5	Stowing	France	Fayol
2600	00.0	13.0	Harmless depth without stowing	France	Fayol
1040	00.0	13.0	Harmless depth with stowing	France	Fayol
390	40.0	7.0	33% of seam put in gob	England	Gresley
325	87.0	30.0		England	Grazebrook
1500	30	5.0	Stowing	England	Hay
600-2400	50			Germany	Menzel

Attempts have been made to formulate rules by which the amount of subsidence may be predicted in advance. Some of the formulæ are based upon the thickness of coal and depth of workings. Most of them include factors for character of rock and filling, but few introduce factors for inclination of the beds.

The discussion of the relation between the depth of workings and the vertical amount of subsidence has brought to the foreground the question as to whether or not subsidence will result irrespective of depth. According to the formulæ of Jicinsky and Menzel there is for each thickness of coal bed a depth beyond which mining will not affect the surface. In 1884 Jicinsky suggested the following:

$$S = m + t - 1.01t \\ = m - .01t$$

in which S = vertical subsidence,
 m = vertical thickness of coal,
 t = thickness of overlying beds.

Menzel suggests the formula

$$S = \frac{t + 350}{350 m}$$

in which S = subsidence in yards,
 t = depth in yards,
 m = thickness of seam in yards.

The factor 350 must be increased to 400 for depths greater than 350 yards. This principle that there is a harmless depth has been supported by Fayol, Banneux, Thiriart, Rziha, Jicinsky, and Menzel.

Fayol formulated two rules as follows:

(1) The height of the zone of subsidence where sandstone predominates and the beds have an inclination less than 40 degrees, and where the area is infinite, does not exceed 200 times the height of the excavation.

(2) When the area is limited, the height of the dome is about twice the breadth excavated for excavations less than 6 feet and up to four times the breadth excavated for seams more than 6 feet.

In general the Germans say that the "dead point" or "harmless depth" has not been reached in Westphalia and question whether or not the term should be used. Callon said that there is no harmless depth, and the majority of the British engineers hold that the removal of all the coal over extensive areas will produce subsidence.*

*The efficiency of filling in reducing subsidence will be considered in Ch. V, see p. 138.

TIME FACTOR IN SUBSIDENCE.

In a study of subsidence it is frequently important to know (1) how soon after the movement shows in the mine workings it will manifest itself upon the surface; (2) the period during which the movement is most severe, and (3) the duration of subsidence.

Upon all of these points there seems to be a great difference of opinion, which is due undoubtedly to the great variety of conditions under which the observations have been made. Fayol wrote, "The period during which movement of the surface may continue is very uncertain. It is allowed to be ten or twelve years in Belgium and at Saarbruck. In other places it has been as long as twenty and even fifty years."*

The committee of the Mining and Metallurgical Society of Ostrau, Moravia, reported in 1881, "The land subsidence manifests itself within one to three months after the collapse observed in the mine. It manifests itself most intensely during the first half year, and then becomes less noticeable. According to our experience it may be assumed that after two years, or more safely, after three years, there do not occur any measureable land subsidences in consequence of a collapsed working."†

S. R. Kay reported that, in working a 5-foot seam at 360 feet, subsidence began about six months after the coal was removed and continued four years.‡

Elevations taken at the Bent Colliery by J. S. Dixon showed that the greater part of the subsidence took place within the first year and that the maximum subsidence came within three years. The depth to the seam was approximately 650 feet.§

In observations made by W. Hay at Shirebrook Colliery, in which mining was being conducted at 1,700 feet, the maximum subsidence appeared in two years.¶

G. E. J. McMurtree reported that the mining of 8 feet of coal at a maximum depth of 800 feet caused subsidence continuing fifteen years.**

In discussing the timbering of roadways in longwall mines in Illinois, S. O. Andros says, "Permanent timbering can be extended

*Colliery Engineer, 1890, Vol. 11, p. 25, 1890.

†Goldreich, p. 63.

‡Proc. Inst. Civ. Eng., Vol. 135, p. 115, 1898.

§Trans. Min. Inst. of Scotland, Vol. 7, p. 224, 1886.

¶Trans. Inst. Min. Eng., Vol. 36, p. 427, 1908.

**Proc. Smith Wales Inst. of Engrs., Vol. 20, p. 367, 1897.

only to that point where the first rapid and violent subsidence has ceased, and it is not usual to extend permanent timbering to any point until the face has been advanced beyond it for at least two years.”*

George Knox says:

“When workings advance rapidly the tendency will be for the strata to bend without fracturing; whereas if the opposite is the case, the force of the motive zone has time to break through, as is frequently shown on the working face after a prolonged stoppage.”†

*Illinois Coal Mining Investigations, Bul. No. 5, p. 32, 1914.

†Knox, George “Mining Subsidence,” Int. Geol. Congress, Vol. 12, p. 804, 1913.

CHAPTER V.

LABORATORY EXPERIMENTS AND DATA.

TESTS AND EXPERIMENTS FOR SECURING DATA.

In the laboratory various experiments and tests can be made to secure data which will be of assistance in the study of subsidence. Among these may be noted the following:

General tests of the materials entering into the problem.

Effect on superimposed material of the removal of part or all of the supports.

Probably the most extensive experiments along this line which have been described in scientific publications have been those made by H. Fayol.*

His experiments to demonstrate subsidence included a variety of materials, as iron, fibre, canvas, rubber, sand, clay, and plaster. He placed iron bars 1.9 inches by 0.19 inch (50 millimeters wide by 5 millimeters thick) one above the other horizontally, the whole being sup-

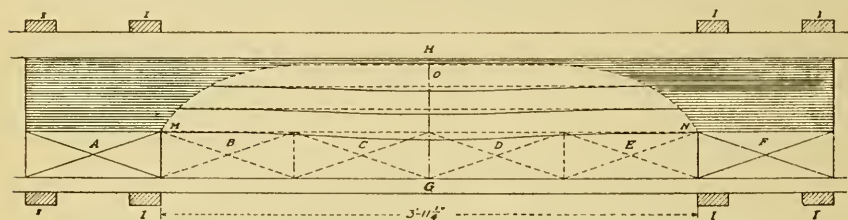


FIG. 50. SAGGING OF IRON BARS.

ported by blocks of wood, *A, B, C, D, E, F*, Fig. 50. These blocks rested upon an iron table *G*. A strong iron rule *H* was placed upon the upper bar of iron, and by means of stays *I*, and bolts, the rule and bars were fastened together and to the table. The wooden blocks *B, C, D, E*, were removed over a length of about 4 feet, and the sagging of the iron bars was noted.

It was found that the deflection of the lower bar was 5 millimeters (0.19 inch), of the tenth bar from the bottom 3.25 millimeters, of the twentieth 1.75 millimeters, and that after the thirtieth bar there

*Fayol, H. "Sur les Mouvements de Terrain Provoqués par l'Exploitation des Mines." *Bul. de la Société de l'Industrie Minérale*. II^e ser., Vol. 14, p. 818, 1885. Translation *Coll. Eng.*, Vol. 11, p. 25, and Vol. 23, p. 548.

was no more bending. The limit of the deflections is the curve *MN* shown in Fig. 49.

The same experiment was tried with flat aloe ropes and with straps of canvas and India rubber in place of the iron bars. With straps of canvas and India rubber the curve of the limits of deflection, that is to say, the limit of the zone of subsidence, had a height nearly equal to the distance between the points of support. This height was about one-third of the same distance for the ropes and one-sixth for the iron bars. Wood and rocks also bend in a manner similar to the materials mentioned.

In order to study the movement in beds of loose materials and in strata that might have been crushed by subsidence, Fayol used artificial beds of earth, sand, clay, plaster, or other materials, and constructed boxes of various dimensions having one side of glass. The box usually employed was 2 feet 7 inches (.80 meter) long, 1 foot (.30 meter) broad, and 1 foot 7 inches (.50 meter) deep. On the bottom of the box were placed, side by side, small pieces of wood of equal thickness, a few centimeters in width, and as long as the breadth of the box. Experiments were made both with one row of these little pieces of wood, and with several placed one above the other. Upon them were laid successive layers of artificial strata, varying from 1 millimeter to several centimeters in thickness. To note the movements, small pieces of paper about $\frac{3}{4}$ inch (2 centimeters) in length and $\frac{3}{8}$ inch (1 centimeter) in width, were put into the planes of stratification, and, on the glass, lines were marked in ink, covering exactly the lines formed by the paper. These lines enabled the least movement to be followed.

By withdrawing the little pieces of wood, excavations were formed and movement produced in the artificial strata.

Fig. 51 represents the movements by taking away, in the order indicated by the numbers, the upper row of wooden pieces, where there were three rows each 0.3937 inch (1 centimeter) in thickness.

The first bed (dry sand), which rests directly on the pieces of wood, falls in as each pillar is withdrawn. The second bed commences to sink only when a certain number of pillars have been taken away. The sinking is shown at first by a slight curve, which has its greatest deflection toward the center of the excavation. Then the third bed follows the second. The movement gradually extends in depth, and reaches the upper bed after the removal of the twelfth pillar. After

the removal of the seventeenth, the beds have become bent, as shown in the sketch, the limits of the deflection being the curves Z_{13} and Z_{17} . (The index figure of the curves is the number of the last pillar taken away; namely, the curves Z_8Z_4 indicate the extent of the movements after the removal of pillars 4 and 8.)

It is apparent that the zone of sinking is a sort of expanding dome, which grows in proportion as the excavation extends.

The bending of the first bed, hardly observable at first, is considerably increased. The second bed sinks rather less than the first, the third less than the second, and the sinking of each diminishes regularly

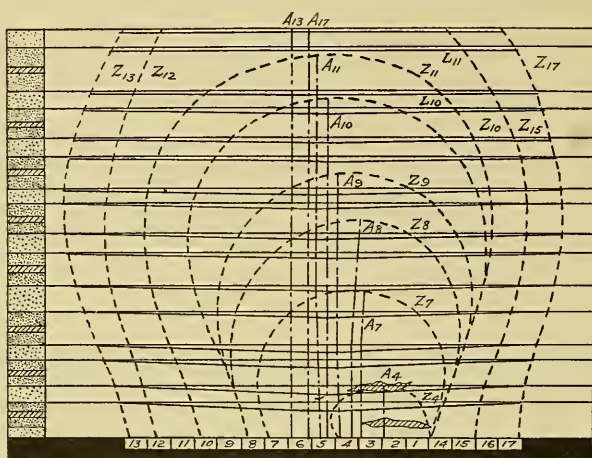


FIG. 51. SUBSIDENCE OF ARTIFICIAL BEDS.

in proportion as it is higher above the excavation. This sinking takes the form of a basin, the center of which is on the vertical axis of the excavation.

The lines A_4 , A_7 , A_8 , A_9 , A_{11} , A_{13} , A_{17} , are lines followed by the greatest deflections of the sunken beds after the removal of the pillars 4, 7, 8, 9, 11, 13, 17. These lines nearly coincide with the axes of the domes, which show the limits of the movement.

Throughout the experiments it was evident after the removal of a certain number of the pillars that the pressure of the superincumbent mass was strong at the center and weak at the circumference of the excavation.

The second row of wooden pillars was taken away and thus the depth of the excavation was doubled. The sinking of the lower beds

increased; some of them fell in; and the broken ground occupied much more space. The disturbance was greater below, but not at the surface. The line of maximum deflection did not remain vertical, and some of the limiting domes were inclined.

Removal of the third row increased the disturbance caused by the removal of the two former; the fractures of the beds and the spaces between the strata were multiplied; some opened more, others closed. As before, the movement started at the lower beds and reached the upper as the excavation extended. The removal of each row of supports results in a new state of stability, which continues if no more pillars are taken away.

Similar experiments were made with beds at various inclinations, and it was found that the line of greatest deflection was between the vertical and the normal, and that it departed further from the normal (that is, the perpendicular to the inclination of the beds) in proportion as the beds became more inclined. Whatever the inclination, the subsidence of each bed had always the form of a basin.

When horizontal beds were covered over by beds dipping at various inclinations; that is, resting unconformably on them, the zone of settlement took the direction of the inclination of the beds and its axis tended to become perpendicular to the beds affected. The lines drawn through the maximum bend of each bed were no longer continuous, but in passing from one set of beds to another were broken and shifted in the direction of the dip of the new set. In all cases the sinking of each bed and of the surface was in the form of a basin.

An experiment was made with horizontal beds, which showed that a block of coal left between two worked-out places may be of no use to protect the surface above it, because the zones of subsidence due to the excavation on either side, which, as already seen, take the form of domes, may overlap each other between the coal and the surface.

As the area of subsidence increases in proportion as the excavation is extended, it may be asked whether there is any limit in depth to the propagation of the movement when the excavation extends indefinitely. To answer this, a mass of horizontal beds was isolated round about by a space being left between them and the vertical sides of the box, and then the wooden pillars (in this case .03937 inch thick) were taken away from under the whole area of the mass. Being entirely free at the sides it might be considered to represent a mass of strata lying over the middle of a working of large extent.

On taking away the pillars, the zone of sinking was seen to increase little by little, and to stop at a certain depth; the movement did not reach the surface. The expansion of the lower beds filled the space excavated and the upper beds rested on the fallen rock. The pressure exerted by the upper strata was very much greater in the middle than at the circumference, and in this case, too, the sinking of the strata was in the form of a basin.

The effect of faults was tested by inserting in a mass of horizontal beds a thin plate of metal, placed at an inclination, and extending the whole width of the beds. This broke the continuity of the beds and represented a fault without throw. Its tendency was to stop the movement from extending above it, though the sinking occurred as usual on its low side, leaving an opening in the plane of the cut, which extended to the surface.

Fayol also made experiments upon the angle of fracture of rocks, the increase in volume of crushed rock, and the compressibility of crushed rock of various sizes.

Effect of Lateral Compression Upon Stratified Materials.

Elaborate experiments were made by Willis* in order to study the deformation of strata by compression. The substance used was beeswax with plaster of Paris to harden it and Venice turpentine to soften it so that by using different proportions of these materials, beds of a wide range of consistency could be constructed. A load of shot was applied upon the beds when constructed, in order to approximate the conditions under which strata at depth are deformed. The machine used for compressing the piles of strata endwise was a massive box of oak provided with a piston which could be advanced by a screw. The pressure chamber was 3 feet $3\frac{3}{8}$ inches long by 6 inches wide. The depth of the box was 1 foot.

T. M. Meade† made a number of experiments, and considered in detail the types of surface which may be developed. He used various kinds and combinations of bars and applied pressure in various ways. An elaborate set of experiments was made to demonstrate circumferential compression. He used for this purpose discs of clay placed within a circumferential band which could be shortened.

*Willis, B. "The Mechanics of Appalachian Structure." 13th An. Rep. U. S. Geol. Sur., Part II, pp. 211-281, 1891.

†Meade, T. M. "Evolution of Earth Structure." p. 146, London, 1903. "The Origin of Mountains," p. 331, London, 1886. Cadell. Trans. Royal Soc. of Edin., Vol. 35, part 7, 1888.

Effect of Vertical Compression Upon Beds of Stratified Materials.

Various tests upon bedded materials used for filling in mines have been made by the United States Bureau of Mines. Incidentally these tests have demonstrated the movement or flow of material in beds



FIG. 52. BENDING OF SHALE UNDER PRESSURE.
(Photo by H. I. Smith, U. S. Bureau of Mines.)

under pressure. Fig. 52 illustrates the bending of shale under pressure in a mine. In this case, however, the bending is accompanied by fracture because of the large movement allowed by the absence of restraint on the under side.

Effect of Lateral Tension Upon Stratified Material.

Not very much work has been done to determine the tensile strength

of rocks and practically nothing has been done upon beds of stratified material.

General Experiments.

General experiments to illustrate geological phenomena and to discover the properties of rocks under conditions of pressure and temperature which may exist at great depths, have been conducted by Daubrée, Adams, and Coker, and various other scientists working at times privately and at other times under the auspices of scientific bureaus of governments and of societies.

The Behavior of Various Types of Artificial Supports.

Extensive tests have been made by the United States Bureau of Mines in various government laboratories and by various mining companies in order to determine the actual and the relative strength of different types of supports.*

SUGGESTED EXPERIMENTS AND TESTS.

(1) In order to study surface subsidence resulting from the removal of supports, it is suggested that a model be constructed, say on a 1/100th scale, both horizontal and vertical, approximating relatively the geological sequence of beds in a given district. The beds should have the same strength relatively in proportion to their weight, or the weight applied, as exists in the geological section which the model represents. The model should be of sufficient extent laterally to represent several panels of a pillar-and-room mine laid out on the panel system. Provision should be made for removing supports so that conditions such as would exist when pillars are drawn may be created.

Observations should be made upon the height of surface from time to time and, after surface movement has ceased, the model should be dissected so that the effects of subsidence below the surface may be noted. Similar models should be constructed to demonstrate working beds of various thicknesses, depths, and dips, and under other systems of mining.

(2) Strength tests of roof materials should be made. The tensile strength and the angle of fracture in bending tests should be determined.

(3) The bending power of the various materials which constitute the mine floor should be measured.

*See Bibliography on Prevention of Subsidence.

(4) In typical mines and under normal working conditions, the pressure or weight of roof should be measured and recorded over as long a period as possible at each point selected.

(5) A study should be made of the composition and physical properties of the rock strata between the beds mined and the surface and also immediately below the beds mined.

CHAPTER VI.

PROTECTION OF OBJECTS ON THE SURFACE.

The surface may be protected by the use of natural or artificial supports. Probably the most general method of preventing subsidence and of protecting objects on the surface is by leaving unmined a portion of the mineral deposit, with the idea that the pillar thus left will have sufficient strength to support the overlying rocks.

In considering the service which a pillar may render and in determining the size of the pillar or other support for protecting specific mine openings or objects on the surface, it will be necessary to consider some of the following factors, and in some cases all of them :

- (1) The unit strength of the material forming the pillar.*
- (2) The height of the mine opening.
- (3) The dip of the mineral deposit.
- (4) The angle of break of the overlying rock.†
- (5) The angle of draw or drag or pull over the pillars, as observed in the district or under similar conditions.
- (6) The strength of the overlying rocks.‡
- (7) The nature and amount of filling in the mined-out area adjacent.
- (8) The depth at which mining may be carried on without affecting the surface.
- (9) The bearing-power of the bottom or floor.
- (10) The weight of overlying materials which must be supported.

To determine the size of pillar necessary to protect mine openings of a given width, it is customary in some textbooks to assume a span of roof and overlying rock to be supported, to estimate the total weight of such a block for the depth of workings, and then, with the known or assumed unit crushing strength of the material to be left in the pillar, the cross-section may be calculated. Such calculations are seldom used in practice and they are open to the objection that they assume a pillar to be uniform throughout, while, as a matter of fact, all bedded deposits are composed of a large number of layers that may vary widely in hardness. For instance, some beds of very hard coal contain

*See p. 70-76.

†See p. 130.

‡See. p. 76.

thin layers of mother coal which reduce the strength of the bed, thus vitiating any calculated results for strength of pillar based on tested specimens taken from the solid part of the bed.

SHAFT PILLARS.

Numerous rules have been formulated for the calculation of shaft pillars in flat seams. Among the best known are the following:*

Merivale.
$$S = 22 \sqrt{\frac{D}{50}}$$

in which S equals length of side of pillar in yards and D equals depth of shaft in fathoms.

Andre. Up to 150 yards deep, a pillar 35 yards square. Up to 175 yards deep, a pillar 40 yards square. Up to 200 yards deep, a pillar 45 yards square, and so on, increasing 5 yards for every 25 yards of depth.

Dron. Draw a line enclosing all the surface buildings, such as engine houses, fans, etc. Make the shaft pillar of such a size that solid coal will be left in around this line for a distance equal to one-third the depth of the shaft.

Wardle. The shaft pillars should not be less than 120 feet square, and the deeper the shaft the larger the pillars. Supposing the minimum to be 120 feet for a depth of 360 feet, 30 feet should be added for every 120 feet in depth.

Hughes. Leave one foot in breadth for every foot in depth; that is, a shaft 600 feet in depth should have a pillar 300 feet in radius.

Pamely. For any depth to 300 feet, it may be sufficient to have a pillar 120 feet square. Adopting this size as a minimum, we may fix any size of pillars for greater depths by increasing the pillar one foot for every four feet in depth.

Foster, R. J. To include the factor of thickness of seam, when conditions are normal, the following formula is suggested:

$$\text{Radius of pillar} = 3\sqrt{Dt}, \text{ in which}$$

$$D = \text{depth of shaft,}$$

$$t = \text{thickness of seam.}$$

Mining Engineering (London). For shallow shafts a minimum of 60 feet radius should be adopted,† and for deeper shafts this should be increased by one-tenth of the depth multiplied by the square root of one-third the thickness of the seam in feet.

*Colliery Engineer, Vol. 17, p. 538, 1897. Coal and Metal Miners' Pocket Book.

†Colliery Engineer, Vol. 18, p. 117, 1897.

$$R = 60 + \frac{D}{10} \sqrt{\frac{t}{3}}$$

Roberton, E. H. In Northumberland and Durham the practice is shown by the following formula:

$$R = \frac{D}{6} + 2\sqrt{Dt}$$

R = radius of the shaft pillar in feet,

D = depth of shaft,

t = thickness of seam.

Scotch engineers, in order to protect buildings have pillars from $1/3$ to $1/5$ larger than the floor plan of the building. This diversity of opinion among engineers is well shown by Fig. 53.*

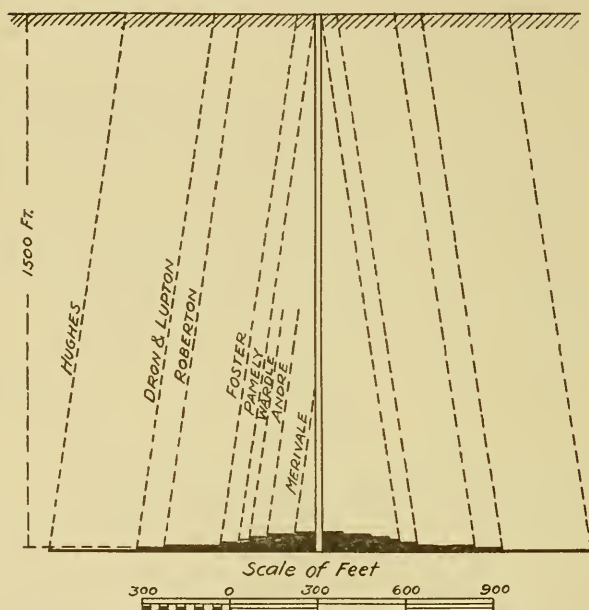


FIG. 53. SIZES OF SHAFT PILLARS ACCORDING TO DIFFERENT FORMULAS.

The Central Coal Basin Rule, presumably founded upon the experience of mining men in Illinois and surrounding states, is: "Leave 100 square feet of coal for each foot that the shaft is deep. If the bottom is soft, the result given by this rule is increased by half. For 5 or 6-foot coal beds, the Central Basin Rule may be used unless it

*Knox, G. Proc. Int. Geol. Cong., Vol. 12, p. 798, 1913.

has been shown by other operating mines in the district that a larger pillar is needed. With thicker coal a larger pillar should be left.”*

The practice of some coal companies in the Connellsville region of Pennsylvania is to leave pillars under buildings so that there is a margin of from 25 to 30 feet of coal around the building. If the tract is large, from 50 to 60 per cent of the coal is removed, the remainder being left in pillars proportioned so that they will serve in the most advantageous way to protect the building. This is the practice for depths from 150 to 300 feet.

In determining the size of the pillar necessary to protect objects upon the surface, as has previously been noted, the ability of the pillar to carry the load is not the only question to be considered. Among the most important of the other problems is that of draw or pull over the pillar previously noted, and the ability of the underlying bed to sustain the load concentrated upon it by the pillar. Quite frequently the underlying bed is less stable and has less crushing strength than the pillar. It seems logical then to proceed as follows in determining the size of pillar necessary to protect an object upon the surface:

- (1) Determine the lateral extent of pillar necessary in order to prevent damage by draw.
- (2) Determine whether the pillar thus outlined is sufficiently large to support, without crushing, the burden of the overlying beds.
- (3) Determine whether the load upon the pillar will cause the pillar to be forced down into the underlying beds, or cause a flow of the underlying material.

ROOM PILLARS.

In his discussion of methods of protecting the surface, M. Fayol referred to the use of pillars between the working places. “The meshes of the network consisting of pillars with working places between them should be made smaller as the workings are shallower. As the depth becomes greater the size of the meshes can be enlarged and dimensions of the areas worked can be increased relatively to the sizes of the pillars that are abandoned, regard being had to the height and width of the zones of subsidence so that the various zones may be kept distinct from each other. This general rule is susceptible of many combinations according to the thickness, the inclination, the number and depth of the seams worked. If the excavation is of small dimensions the subsidences which take place above them are restricted in size and become

*Illinois Miners' and Mechanics' Institutes, Instruction Pamphlet No. 1, p. 49.

enlarged both in width and height as the excavation increases in area. If each of the pillars, 1, 3, 5, and 7 (Fig. 54) be taken out singly, zones of subsidence similar to Z_1 , Z_3 , Z_5 , and Z_7 , would be produced; but when pillar 2 is taken out the line of roof subsides on to the floor, and the zone of subsidence rises to Z_2 . The same thing happens when No. 6 pillar is taken out, and if No. 4 pillar is taken out, the space comprised between the zones Z_2 and Z_6 is set in motion and determines the formation of zones Z_4 .*

It follows from this statement of Fayol that if the room pillars are properly proportioned and properly spaced, the disturbance of the strata may be limited to the volume within the zones. The material

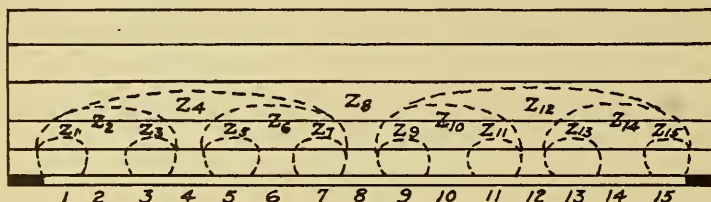


FIG. 54. EFFECT OF EXTENT OF EXCAVATION ON AMOUNT OF MOVEMENT.

outside these zones throws no weight upon the material within the zones. Necessarily, then, any vertical pressure must fall upon unmined material forming the pillars and the pillars must be large enough to withstand the pressure.

In a paper before the Pennsylvania State Anthracite Mine Cave Commission, 1913, Douglas Bunting said: "The application of a formula for determining the safe size of coal pillars for various thicknesses of veins and depths can be considered practical for depths greater than 500 feet, but it is doubtful if the same formula would be of any practical value for application to veins at less depth and certainly of diminishing practical value with reduction in depth and thickness of veins for the reasons that the variable conditions of vein, top, bottom, etc., are of more consequence with small pillars than with large pillars."†

D. Bunting‡ made a careful study of chamber pillars in deep anthracite mines on light dips. He considered the crushing strength of coal which for anthracite was found to average 2,500 pounds per square

*Proc. South Wales Inst. Eng., Vol. 20, p. 340, 1897. It should be noted that these zones outline the dome through which the movement extends, and not the limit of the *falling zone*, as described by Rziha.

†Bunting, D. "Pillar and Artificial Support in Coal Mining, With Particular Reference to Adequate Surface Protection." Pa. Legislative Journal, Appendix, Vol. 5, p. 5988, 1913.

‡Bunting, D. "Chamber Fillers in Deep Anthracite Mines." Trans. A. I. M. E., Vol. 42, p. 236, 1911.

inch for cubes. The ratio between the strength of prisms and cubes was taken as follows:

$$\frac{\text{Strength of prism}}{\text{Strength of cube}} = 0.70 + 0.30 \frac{b_1}{h}$$

in which

b_1 = width of pillar,

h = thickness of vein.

The weight of overlying strata was taken at 144 pounds per cubic foot.

$$(1) \text{ Load per square foot on a pillar} = \frac{144 y z b_1}{b_1^2}, \text{ in which}$$

y = depth below the surface,

b_1 = width of pillar,

z = distance between chamber centers.

With 1,000 pounds per square inch as the safe load for a cube we obtain by substituting in equation (1):

$$\frac{144 y z b_1}{b_1^2} = 144,000 \left(0.70 + 0.30 \frac{b_1}{h} \right)$$

$$\text{or } y z = 1,000 \left(0.70 + 0.30 \frac{b_1}{h} \right) b_1$$

By making proper allowance for the crushing strength of the pillar material and the weight of overburden, this formula may be used generally for flat beds.

The relative widths of rooms and pillars are determined largely by practice. For bituminous coal of medium hardness and good roof and floor, the following rule is sometimes used: "Make the thickness of room pillars equal to one per cent of the depth of cover for each foot of thickness of the seam, according to the expression:

$$W_p = \frac{tD}{100}, \text{ in which}$$

W_p = pillar width,

t = thickness of seam,

D = depth of cover,

and then make the width of room or opening equal to the depth of cover divided by the width of pillar thus found, according to the expression:

$$W_o = \frac{D}{W_p}$$

in which W_o is the width of room.

"Frail coal and coal that disintegrates readily when exposed to the air, and a soft bottom, may increase the width of pillar required as much as 50 per cent of the amount found above; also, a hard roof may increase

the same as much as 25 per cent; while, on the other hand, a frail roof or a hard coal or floor may reduce the width of pillar required 25 per cent.”*

“As to the thickness of pillars in the Pittsburgh seam with strata 100 to 500 feet thick, the following rule should be a safe one to follow, in which the pitch is from 1 to 5 per cent:

Thickness of Surface Feet	Thickness of Pillars George's Creek Feet	Thickness of Pillars Fairmont Feet
100	25	18
150	32	20
200	40	25
250	50	30
300	60	35
350	70	40
400	80	45
450	90	50
500	100	55

These figures are based on experience in this seam, where the floor or bottom is hard and not affected by water. For a fireclay bottom somewhat thicker pillars would be necessary to withstand any extraordinary weight. Rooms should be not more than 14 feet in width in the Georges Creek region and 20 feet in the Fairmont region.”†

The average dimensions of pillars and rooms in ordinary pillar-and-room mining in Illinois are shown in Table 13.‡

TABLE 13.
DIMENSIONS OF PILLARS AND ROOMS IN PILLAR-AND-ROOM
MINING IN ILLINOIS.

District	Average Depth in Feet	Room Width in Feet	Pillar Width in Feet	Average Thickness of Coal in Feet
II	140	26	19	{ top bench 2 ft.
III	90	22	18	{ bottom bench, 3 ft. 9 in.
IV	201	25	9	4 ft.
V	243	26	16	4 ft. 8 in.
VI	270	22	18	4 ft. 8 in.
VII	227	31	30	9 ft. 5 in.
VIII	174	27	8	7 ft.
Average of 48 Representative mines	208	26	19	{ Seam No. 6—6 ft.
				{ Seam No. 7—5 ft.

The question of the thickness of cover is an important one in connection with the size of the room pillars and particularly when the draw-

*Coal and Metal Miners' Pocket Book, 9th Ed., p. 286, 1907.

†Reppert, A. E. "Pillar Falls and the Economical Recovery of Coal From Pillars." W. Va. Coal Min. Inst., p. 116, 1911.

‡Ill. Coal Min. Investigation, Bul. No. 13, p. 76, 1915.

ing of pillars is considered. This has been emphasized by F. W. Cunningham as follows: "The topography of the surface relative to hills and vales should be considered when starting to draw pillars and relative to this subject a question may be asked, which is an important one, viz., How many coal properties have contour maps of the surface? Suppose, for example, the rocks at the surface rise abruptly on each side of a narrow valley to say 200 or 300 feet. Would it be proper to commence pillar drawing under this valley?"*

STRENGTH OF ROOF.

In determining the limits of mining under heavy wash, D. Bunting considered the strength of slabs of roof rock supported by pillars. "In deriving a formula for computing the breaking load of a slab of stone

from the formula $\frac{pI}{e} = M_m$, let W represent the distributed loading plus the weight of the beam itself in pounds, b , d , L represent the breadth, depth, and span, respectively, in inches, and R equal the modulus of rupture in pounds per square inch.

Bunting suggests that "the modulus of rupture does not express the actual stress in the extreme fiber of the beam of rock, but is a quantity useful only as a basis of comparison." He gives the following safe unit stresses for stone, recommended by W. J. Douglas as illustrative of possibly a fair average of safe stresses:

	Compression Lbs. per sq. in.	Shear Lbs. per sq. in.	Tension Lbs. per sq. in.
Granite	1200	200	150
Limestone	800	150	125
Sandstone	700	150	75

The maximum bending moment for a constrained or prismatic beam is equal to $\frac{WL}{12}$. By substituting in the formula for flexure ($\frac{pI}{e} = M_m$) we obtain the formula $W = \frac{2bd^2}{L}R$. Likewise, the maximum moment at the center of such beam being equal to $\frac{WL}{24}$, the formula becomes $W = \frac{4bd^2}{L}R$.

It is evident that failure of flexure would theoretically take place at the points of support and not at the center of the span.

*Cunningham, F. W. "Methods of Removing Coal Pillars." Proc. Coal Min. Inst. of America, p. 35, 1911.

In applying the formula $W = \frac{2bd^2}{L}R$ to the case of a slab spanning a breast or other mine opening, the weight of the overlying material will be taken at 108 pounds per cubic foot, and the depth of the opening below the surface will be designated by d' in feet.

Then, $W = \frac{108Ld'}{12}$, which would be the loading of the slab with a breadth of 1 foot. Substituting this value of W in the equation $W = \frac{2bd^2}{L}R$ and simplifying, the equation $d^2R = \frac{3}{8}L^2d'$ is obtained. If, however, the weight of the overlying material per cubic foot be represented by w , the expression becomes $d^2R = \frac{wL^2d'}{288}$

In the use of the formula derived for determining the minimum safe thickness of rock over mine openings for various depths below the surface, consideration must be given to a number of conditions, the more important of which are:

1. Nature of the top immediately above the coal seam and its comparative strength and liability to disintegration upon exposure to the atmosphere.

2. Nature and thickness of the bed, the ability of the pillars to resist squeezing, and the liability of disturbance of the overlying strata, due to covering or squeezing in underlying beds.

3. Probable errors in relative vertical location of top of rock and mine workings.

4. Possibility of the existence of deep gorges and pot holes.*

In order to arrive at a brief solution in calculating pillars of quartzite for Rand mines, Richardson† made use of the following formulæ:

Bending

1. $F_b = 106 \frac{Kt}{l^2w}$

2. $L = 10.2 \sqrt{\frac{Kt}{w}}$

3. $W = 106 Kt^2 - l^2tw$

Shearing

1. $F_s = \frac{34.2 dk}{l^2w}$

2. $L = 5.85 \sqrt{\frac{dk}{w}}$

*Bunting, D. "Limits of Mining Under Heavy Wash." Amer. Int. Min. Engrs. No. 97, p. 18, 1915.

†Richardson, A. "Subsidence in Underground Mines." Eng. and Min. Jour. Vol. 84, p. 196, 1907.

$$3. \quad W = (34.2 dk - l^2 w) t$$

in which F_b = factor of safety for bending,

F_s = factor of safety in shearing,

l = length of side of slab or distance from center to center of pillars,

L = length of side of a slab which will only support its own weight,

W = total distributed load which the slab will carry in addition to its own weight,

K = compressive strength of pillar material pounds per square inch,

t = thickness of slab in feet,

w = weight of a cubic foot in pounds,

d = diameter of pillars in feet.

He presumes that "where the slope areas are not extensive the weight of the upper masses will be supported by their own strength," and calculates the size of pillar which will support continuous slabs of rock, homogeneous and uniformly loaded. By use of the formula he prepared a table of sizes of pillars for various spaces and concluded that slabs usually break up by shearing and that the strength to resist this depends on the size and distance apart of supports.

FILLING METHODS.

Various materials and methods are employed to protect the surface if it is deemed advisable to remove all the material deposit, or if the material left in the forms of pillars is found inadequate to support the surface.

Waste material resulting from the regular mining operations or broken for this particular purpose may be stowed or packed into the excavation. If sufficient or suitable material is not available under ground it may be lowered or dropped from the surface and stowed where needed. Crushed materials may be introduced from the surface and transported through pipes and stowed by water or compressed air. Timber, steel, or various forms of masonry may be employed to support areas upon which important structures may be erected.

This entire subject has been studied by the engineers engaged upon investigations of subsidence and surface support in the Pennsylvania Anthracite field, who say:

"Most coal beds consist of interstratified layers of coal, fireclay, slate, and bony coal, the latter three composing the principal refuse material of the mine. In these beds, in which it is necessary to remove some of the roof rocks or take up some of the floor of the mine in order to obtain

height sufficient for the mules and the men to travel along the roads, much mine refuse is produced, which is stored in the chambers. In beds less than four feet thick many chambers are filled with mine refuse or gob from floor to roof. In places this gob is merely thrown in carelessly or is shoveled in; in other localities it is packed as tightly as possible by hand. When there is much interstratified fireclay or bone in the coal beds there will be larger quantities of the gob, and the thinner the bed the greater will be the quantity of mine rock raised or taken down for roads. The supporting value of stored gob depends upon the compressibility of the material of which it is composed.”*

Griffith's Method of Filling.—It has been suggested by William Griffith that worked-out portions of mines be filled by blasting up the bottom and shooting down the roof. The suggestion was made in connection with a report to the Scranton Mine Cave Commission and Mr. Griffith has secured patent (U. S. Patent 1,004,418) covering this method. W. Griffith and E. J. Conner say: “It is a well known fact that loose rock occupies from $1\frac{2}{3}$ to twice the volume of the same weight of rock in place. Your engineers have conceived the idea of taking advantage of this fact, well known to engineers, for the purpose of cheaply producing an adequate support of the rock and surface above certain classes of coal beds under the city of Scranton. So far as we know, this method, in its entirety, has never been used before in any coal mining district, and the suggestion is here made for the first time.

“The process is applicable to beds less than 6 feet in thickness and consists simply in blowing up the floor and shooting down the roof of the mine, each to a depth equal to the thickness of the coal bed. This produces a total thickness of loose rock equal to three times the thickness of the coal. The rock would be well packed together and have great supporting power, and, moreover, the desired ends would be attained in a comparatively inexpensive manner.†

The method of blasting stowing material from the hanging or foot walls is commonly used in metalliferous mines.

GOB STOWAGE IN LONGWALL MINING.

In longwall mining “the rock obtained from brushing the roof, that which remains after building the pack walls, and the clay obtained from undermining the coal are thrown behind the pack walls lining the roads.

*Griffith, Wm., and Conner, Eli T. “Mining Conditions Under the City of Scranton, Pa.” U. S. Bureau of Mines, Bul. No. 25, p. 52, 1912.

†Griffith, William, and Conner, Eli J. “Mining Conditions Under the City of Scranton, Pa.” U. S. Bureau of Mines, Bul. No. 25, p. 57, 1912.

The gob area is usually filled with rock and clay to within 2 to 5 feet of the coal face. This loose rock and clay helps to support the roof and control the roof weight on the coal face. The waste should fill the gob sufficiently to allow the roof to come down gradually without breaking off short at the face of the pack walls, but should not fill the gob so completely that it carries too much of the roof and does not throw enough weight on the face of the coal. The width of the pack wall, called 'building,' necessary to prevent the walls from squeezing out and filling the roadway when the roof weight comes on them depends upon local conditions. The Third Vein District Agreement between the Illinois Coal Operators' Association and the United Mine Workers of America provides: "The miner shall build 4 yards of wall at each side of his road, and if he has more rock than is required therefor he shall not load any of it until he has filled his place therewith." Room centers at the long-wall face (in Illinois) are usually 42 feet apart.**

GOB PIERS.

In some cases, especially when the prevention of any movement of the surface is especially desirable, gob piers are used. These are pillars of waste rock, either laid up by hand throughout, or having the outer wall carefully laid while the interior is filled with refuse shoveled in. The resistance of such supports to compression depends upon the material used and the care with which they are built.

CONCRETE AND MASONRY PIERS.

These forms of support are more expensive than those previously mentioned and are likewise more substantial. Masonry has frequently been used to support the roof below important structures and occasionally to support the walls of inclined beds and the overburden.

One of the earliest and also one of the most notable examples of the extensive use of masonry in metal mines was the construction at the Tilly Foster Iron Mines.† The total masonry constructed amounted to 20,189 cubic yards.

Whenever possible the concrete used is introduced from the surface through boreholes. An interesting example of such use of concrete is reported by Mr. Temple Chapman of Webb City, Missouri. In a zinc mine six concrete piers were constructed, 35 feet high by 16 feet wide and

*Andros, S. O. "Mining Practice in District I." Illinois Coal Mining Investigations. Bul. No. 5, p. 20, 1914.

†Engel, L. G. "Masonry Supports for Hanging Walls at the Tilly Foster Iron Mines." Columbia School of Mines Quarterly, Vol. 6, p. 289, 1885.

20 feet long. The measures were horizontal and the distance from the surface to the roof was 150 feet. First a 6-inch hole was drilled from the surface to the roof with a churn drill at a cost of \$0.90 per foot. A large pile of tailings was close at hand, consisting of crushed rock passed through a half-inch hole and containing some finer material and sand. The mixture was six parts of tailings to one part of cement, which is about equal to four parts of gravel, two of sand, and one of cement. This was mixed mechanically and discharged direct from the mixer into the drill hole. Underground two men were kept busy building up the form, which was made of 1 by 12 inch board laid on edge and 2 by 6 inches set vertically at 2-foot intervals and wired together across through the form. Worn perforated trommel screen jackets cut in strips 10 feet long by 4 inches wide were used to reinforce the concrete. These were laid east and west a foot apart and the concrete was poured. A foot higher similar strips were placed at right angles to the first, and so on. A few 60-pound rails were put into the tops of the piers, projecting from pier to pier where possible. These piers were placed between ore pillars, the plan being to remove these ore pillars. The piers were built at a cost of \$3 per cubic yard at a time when the ore in the pillars was worth \$12 per cubic yard.*

A novel method of using concrete in connection with packing or stowing was employed in France and reported by J. H. Piffaut.† The coal bed, quite thick and highly inclined, was worked in 8-foot slices in descending order. Upon the floor of a slice was spread a layer of coal dust from 1 to 1½ inches thick; then a layer of concrete from 8 to 10 inches thick; and upon this was placed the ordinary packing. As the working place had previously been timbered, the concrete surrounds the base of the posts. When the next slice is removed the concrete floor of the upper slice acts as a roof for the lower slice, which is timbered in the regular manner in order to support the concrete loaded with packing. It is claimed that this has proved satisfactory in the mining of thick beds.

COGS.

Cogs are cribs of timber filled with waste rock. They may be erected quickly and they have great strength. They find some use in the ordinary course of mining, but they are especially useful in preventing an impending squeeze, or in stopping one that has already started by

*Correspondence.

†Piffaut, J. H. "The Use of Cement-Concrete in the Working of Thick Coal Seams." Trans. Inst. Min. Eng., Vol. 29, p. 274, 1904.

supplying such support that the overlying strata break through to the surface. Their strength is, of course, lost when the timber decays.

SPECIAL TYPES OF COGS AND PIERS.

William Griffith has recently developed a cog which it is expected will be many times as strong as the ordinary timber cog and both stronger and more durable than the common concrete pier. The objection to concrete cogs or piers is that when the compressive strength is exceeded the mass of concrete will go to pieces and will give no support whatever. With rock and timber piers, even though the percentage of compression may be large, the piers do not go to pieces but have some supporting power. The concrete pier will collapse suddenly while the other types of piers will be gradually deformed. Mr. Griffith says that what is needed is something that will bear up under the heaviest weight, that will "give" to a certain extent and will then withstand the continuing burden. In his new pier, concrete is the basic material with timber to reinforce it. The piers are constructed so that it is impossible for the timber to pull away and for the concrete to be crushed. "The timber should be creosoted and after the pier is constructed it should be coated on the outside with cement by the use of the cement gun."

Tests show that a cog or pier, forty days old, will sustain for each square foot of horizontal area:

- 7 tons with a compression of 1 per cent.
- 130 tons with a compression of 3 per cent.
- 140 tons with a compression of 14 per cent.

IRON SUPPORTS.

From time to time various types of metal supports have been tried in the working places of mines. Where iron props or posts have been installed in the Scranton district no subsidence occurred and it is the opinion of the local engineers that the effectiveness of such props has not been demonstrated. Rolled steel shapes are being quite extensively used as legs and collars and as beams for the support of wide openings, such as shaft bottoms. Iron supports have also been tried in metalliferous mines, but, except for the support of the shafts, stations, and passageways, they have never found extensive application. Iron props have been used in foreign mines.

HYDRAULIC FILLING.

One of the most important methods of protecting the surface above

mine workings is by filling the workings with fine material carried by water through pipes. In his report upon this method as used in the Pennsylvania Anthracite fields, CharlesENZIAN says: "Heretofore the process has been termed and even at present is known in the Pennsylvania Anthracite region as 'slushing,' 'flushing,' and 'silting.' As a result of various suggestions from men of long experience in this work, the name 'hydraulic mine filling' was adopted for the use of the report."* The process has been used in (a) extinguishing mine fires, (b) arresting mine squeezes, (c) supporting the surface, (d) reclaiming pillars and increasing the yield of coal, (e) disposing of spoil banks, and (f) in lessening stream pollution.

According to the Colliery Engineer, Vol. 33, p. 537, flushing was first used August, 1884, by John Veith, General Inside Superintendent of the Philadelphia & Reading Coal and Iron Company, who employed it to extinguish a fire in the Buck Ridge slope near Shamokin, Pennsylvania.

The second use of flushing and its first use to support or control overlying strata is credited in the same reference to Frank Pardee of Hazleton, Pennsylvania. In 1886 F. Pardee used the system to stop a squeeze which threatened the slope and breaker of the Laurel Hill colliery at Hazleton. He accomplished this by flushing adjacent breasts with culm. The breasts were steeply pitched. The squeeze was stopped by means of natural pillars, each 10 yards wide, and two breasts filled with culm, each 10 yards wide, and the subsiding rock broke off.

The most extensive early use of flushing was at the Kohinoor colliery at Shenandoah, Pennsylvania. When this colliery was taken over by the Philadelphia & Reading Coal and Iron Company, January 1, 1884, it was found that because of workings in the thick Mammoth seam a large part of the town of Shenandoah was likely to be affected by a subsidence of the surface. The Mammoth seam was from 40 to 60 feet thick, thus making timbering impossible. The coal was about 400 feet from the surface. After various methods had been suggested, the officials of the company decided to flush culm into the workings, none of those engaged in the enterprise knowing of the previous use of culm for roof support by F. Pardee.

A very detailed description of the method used in flushing the culm into the workings can be found in the above reference in the Colliery Engineer and in Bulletin No. 60 of the U. S. Bureau of Mines.

The materials that have been used or may be available for hydraulic

*ENZIAN, Charles "Hydraulic Mine Filling." U. S. Bureau of Mines, Bul. No. 60, 1913.

mine filling include culm, ashes, crushed refuse from coal washing plants, sand, gravel, clay, loam, granulated slag, and crushed rock.

The methods employed and results accomplished have been described by Davies, Griffith, and Enzian.*

The process consists of conveying culm, sand, ashes, etc., to the desired place by means of water, the method used depending upon conditions. If the pipe line can be laid on a steep grade from end to end, the material will flow easily and little water will be required. On the other hand, if the grade is light or if it must be reversed over part of the line a larger quantity of water is required and, of course, a larger pipe. There must always be sufficient velocity to prevent settling of the solids and this can be obtained only by having sufficient head. Naturally the whole operation is easiest when the grade is steep, the pipe short, and the curves and connections few.

To avoid blockage of the pipe, clear water should be allowed to flow for a few minutes before filling is added, in order to establish a current throughout the pipe and when the flushing is to be interrupted, the addition of filling should be stopped some time before the water is shut off, so that the solid matter may be washed out of the pipe.

The proportion of water required depends upon the velocity of the current and the nature of the filling material. In general practice about 90 per cent of the material carried by the line is water.†

Good practice requires absolute control of the filling until it is deposited at the desired place. This necessitates carrying the pipe line to the place of deposit, no allowance being made for flow in chambers.

As the filling should be interrupted after 200 to 400 cubic yards have been deposited and the material be allowed to settle for fifteen to eighteen hours, it is desirable that branch lines be laid to different points, so that the process, as a whole, need not be interrupted. During the period of settling, water seeps out and the material shrinks from 1 to 10 per cent in volume. It is necessary that the drainage be so controlled that the least possible solid matter will be carried away. The finest part of the filling has an important part in the cementation of the mass.

The process requires careful and continuous attention, though the number of men employed need not be large. Generally, there should

*Davies, J. B. "Flushing Culm." *Mines and Minerals*, Vol. 18, pp. 342, 389, 1898.
Griffith, William. "Flushing Culm." *Mines and Minerals*, Vol. 20, p. 388, 1900. Enzian, Charles "Hydraulic Mine Filling." U. S. Bureau of Mines, Bul. No. 60, pp. 58-60.

†Wilson, H. M. "Irrigation Engineering." Pp. 61-69, 338, 344, Revised edition, 1910.

be one man for the surface, one to patrol each 1,000 feet of pipe line, and one to inspect the filling.

The results obtained have been very satisfactory and a large amount of material formerly deposited on the surface is now washed back into the mines.

In discussing wastes in Illinois coal mining, G. S. Rice commented upon the feasibility of employing hydraulic fillings. He noted the use of culm for filling in Pennsylvania and stated that "In Illinois, the substitute would have to be surface sands and gravel. That this would be impracticable in the great majority of cases throughout the State is self-evident, particularly if water, the usual vehicle for transportation, is employed, inasmuch as the majority of the thick seams in Illinois have clay under them which water would soften, and thus tend to cause a 'squeeze.' Aside from this, much farm land would be destroyed in getting the filling material."*

In longwall mining the application of hydraulic filling under present practice does not seem to be generally feasible. Hydraulic filling in flat seams worked on the longwall plan was inaugurated near Liege, Belgium, in 1913, but has not been employed on a sufficiently large scale to justify a statement that it is practicable for flat seams.†

Over a hundred collieries in Upper Silesia have employed hydraulic filling‡ in seams varying from 4 to 40 feet in thickness. Subsidence has been reduced from 30 to 70 per cent to 0.3 to 7.8 per cent of the height of the seam. In 1914 twenty-seven collieries, employing forty equipments, used hydraulic filling. The sand commonly used in Silesia for filling is mined with steam shovels and then transported by railroad, sometimes for considerable distance, to the mine, where it is dumped on a grizzly to remove the boulders and then mixed with a suitable amount of water to flush it into the mine. At one mine, at least, the boulders are crushed and mixed with sand filling. A detailed description of the methods used in Upper Silesia will be found in the reports of the Upper Silesia Mining Association. In the Saarbrücken district there are on state-owned lands more than twenty independent hydraulic-filling installations, costing \$350,000. This method is employed for iron and potash mines as well as in the coal mines.

"The only fairly extensive installations at work in Britain is that

*Rice, G. S. "Mining Wastes and Mining Costs in Illinois." Ill. State Geol. Sur., Bul. No. 14, p. 220, 1909.

†See Trans. Inst. Min. Eng., Vol. 46, p. 439, 1913-1914.

‡Trans. Inst. Min. Eng. Vol. 46, p. 534, 1912. Report of British Consul-General, Westphalia, p. 25, 1911.

of the Wishaw Coal Mining Company, Motherwell. There are other installations in a small form, or under consideration, but nothing yet has been adopted on an extensive scale. A small trial outfit has been installed at one of the Fife pits, and there is a proposition to use hydraulic stowing where the seams run under the sea. There is a small installation at the Crowgarth iron-ore mine.”*

“In France it has been used, especially at the collieries in the Department of the Pas-de-Calais, and also in the coal fields of St. Etienne. In Belgium it is used at several collieries. In Spain the Penarroza Colliery is erecting a plant, and several collieries in Austria, as well as Poland and Russia, are employing the system. It is used also at lignite mines in Manchuria and in the gold mines of Australia and the Transvaal.”†

Gullachsen reports that in order to avoid the great expense of pumping to the surface the water used in hydraulic filling the Cinderella Deep mine introduced a system by which sand is sent into the mine in a dry condition. A wooden box-laundry was constructed measuring 12 by 11 inches in inside cross-section. This laundry was carried down the vertical shaft to a depth of 3,900 feet to the level at which the filling material was required. The sand, which should not contain more than 7 per cent of moisture, is stored in a surface bin, from which it is taken on a conveyor belt to the top of the shaft and there discharged into the laundry. On reaching the bottom of the laundry, it falls on a steeply inclined iron plate, at which point jets of water are turned into the sand, which is then carried away as a pulp. The great objection to this system is the difficulty of securing a constant supply of dry sand. As soon as the sand contains more than 7 per cent of moisture, it is inclined to adhere gradually to the sides of the laundry, which in time becomes choked. The laundry was connected to a Roots blower and jets of compressed air introduced, the idea being to assist the drying of the sand and to increase the velocity of the falling stream, but this device was found to result in only a very slight improvement.”

PNEUMATIC FILLING.

The stowing of crushed rock by means of compressed air has been successfully employed in the Lake Superior copper district at several

*Paton, J. D. “Modern Developments in Hydraulic Stowing.” Trans. Inst. Min. Engrs., Vol. 47, p. 468, 1914. See also Trans. Inst. Min. Engrs., Vol. 48, p. 134, 1914, and Iron and Coal Trades Review, Vol. 89, p. 454, 1914.

†Gullachsen, B. C. “Hydraulic Stowing in the Gold Mines of the Witwatersrand.” Trans. Inst. Min. Eng., Vol. 48, p. 122, 1914. See also Trans. Inst. Min. Eng., Vol. 41, p. 586, 1910, and Trans. Inst. Min. Eng., Vol. 43, p. 632, 1911.

mines, having been developed at the Champion mine of the Copper Range Company by F. W. Denton. Stamp sands or tailings from the concentration plant are hauled in railroad cars a distance of eighteen miles and discharged through pipes into the worked-out stopes. It is claimed that by the use of this material a saving is made over the method of support formerly used. In order to provide sufficient material for filling the stopes, waste rock secured from sorting in the stopes was supplemented by rock blasted out of the walls. At present the sand is used in addition to the waste material discarded in the stopes.*

SUPPORTING POWER OF FILLING.

The problem of support of surface by filling suggests two important points, in addition to the controlling factor of the cost of filling. When the worked-out portions of the mine are filled by the natural process of caving, the factor of increase of volume of material should be known. Moreover, as the overlying beds sink upon this filling the factor of compressibility of the filling must be considered. Fayol made extensive and careful investigations along these lines, and his determinations of the increase of volume are shown in Table 14.

TABLE 14.

INCREASE IN VOLUME OF MATERIALS IN FILLING.

Relative Volumes

Nature of Rock	Unbroken	Crushed to Powder	Grains .078 to .118 inch (2-3 mm.)	Grains .393 to .59 inch (10-15 mm.)	Grains .59 to .787 inch (15-20 mm.)	Mixtures, Grains and Fine Dust
Clay	100	196	209	226	225	216
Shale	100	213	210	221	224	229
Sandstone ...	100	219	214	211	210	214
Coal	100	207	224	199	223	202

The mixture of large and small pieces of sandstone and shale commonly used for stowing increases in volume about 60 per cent. The greater the increases in volume, the more easily is the crushed material compressed. Fayol's results of tests of compression upon crushed material are given in Table 15.

The pressures noted in Columns I, II, III, and IV correspond to depths of strata of 1,638, 3,276, 8,190, and 16,380 feet, respectively.

*"Sand Filling at Champion Copper Co., Painesdale, Michigan." Min. and Eng. World, Vol. 41, p. 1194, 1914.

TABLE 15.

RESULTS OF TESTS OF COMPRESSION UPON CRUSHED MATERIAL.

Nature of Rock	Space Occupied Before Being Broken	Rocks Having Been Previously Crushed or Broken, the Space Occupied under Pressure of			
		I. 1422 lb. per sq. in. 100 kgm. per sq. cm.	II. 2844 lb. per sq. in. 200 kgm. per sq. cm.	III. 7110 lb. per sq. in. 500 kgm. per sq. cm.	IV. 14,220 lb. per sq. in. 1000 kgm. per sq. cm.
Clay	100	100	90	75	70
Shale	100	128	116	110	97
Sandstone	100	136	125	120	105
Coal	100	130	125	118	109

Fayol concluded that the material which ordinarily fills the goaves of mines always occupies a larger space than it did originally, and after an expansion of about 60 per cent it appears to undergo in workings of from 300 to 900 feet in depth a compression of about 30 per cent, which leaves a volume about 12 per cent larger than the volume of the unbroken rock.*

The supporting strength of dry filling as studied in connection with the problem of surface support at Scranton, Pennsylvania, is shown in Table 16.†

Anton Frieser reports that in coal mining in Bohemia hydraulic filling has been carried on extensively and that, with such filling at depths of from 60 to 200 feet, the roof pressure compresses one volume of ordinary stone-and-sand packing to 0.6, clay packing compresses to 0.5, and puddled sand and ashes to 0.8 or 0.9.‡

In the Ruhr coal district of Germany, filling has been used extensively and the amount of compression has been noted carefully.§ This has been possible as new openings were driven through workings which had been filled from two to eight years previously.

Dr. Niesz has found that gobbing, under pressure, may lose four-tenths of its height, small-grained pit-heap material 25 per cent, and pure loose sand 8 per cent.¶

The commission reporting upon the slide at Turtle Mt., Frank, Alberta, Canada, commented upon the efficiency of various kinds of filling in mine workings. The general statement was made that under

*Colliery Engineer, Vol. 33, p. 548, 1913.

†U. S. Bureau of Mines, Bul. No. 25, p. 59, 1912.

‡Oestr. Zeit für Berg-und Hüttenwesen, Vol. 43, p. 253, 1895.

§Oberhausen, J. "Compression of Slope Fillings," Glückauf, p. 1146, Nov. 22, 1902.

¶Translation in Columbia School Mines Quarterly, Vol. 26, p. 271, 1904.

§Zeit für Berg-, Hütt-, u.-Salinew., Vol. 58, p. 418, 1910.

TABLE 16.
SUPPORTING STRENGTH OF VARIOUS FORMS OF DRY FILLING.

Kind of Material Comprising the Artificial Supports	Approximate Depth, in Feet, of Column of Coal Measure Rock, 1 Foot Square, Necessary to Compress Artificial Roof Support					
	1	3	5	10	20	30
Per Cent. of Compression						
1. Rectangular gob piers, ordinary construction	Feet	Feet	Feet	Feet	Feet	Feet
2. Circular piers of mine rock, well constructed.....	10	12	36	125		*306
3. Timber cogs filled with gob, average construction.....	46	75	146	292		*512
4. Loose pile of broken sandstone through 1¾-inch ring, 40 per cent. voids.....	8	68	182	270		*419
5. Pile broken sandstone, 40 per cent. voids, voids filled with sand		20	53	124		*298
6. Loose pile large size broken sand rock, 45 per cent. voids.....		21	53	186		*465
7. Mine room filled with large broken sand rock, 50 per cent. voids.....	48	66	121	351		*492
8. Mine room filled with broken sandstone, 40 per cent. voids.....	12	27	45	117	434	a615
9. Mine room filled with broken sandstone, 40 per cent. voids filled with sand.....	44	74	177	619		1,310
10. Mine chamber filled with dry coal ashes, 64 per cent. voids.....	46	77	325	6,000		b8,860
11. Mine room filled with dry river sand	13	25	70	143		332
12. Mine room filled with river sand flushed in with water..	12	40	70	442	1,715	6,649
13. Mine chamber filled with coal culm flushed in with water..	111	522	891	2,310		c8,860
14. Concrete pier, 1 part cement, 7 parts sand and gravel; 5 months old.....	32	118	190	472	1,822	5,965
	117	1,092	(c)			
Resistance of flushed culm.....	1.0	1.0	1.0	1	1	†.....
Resistance of flushed sand.....	3.5	4.4	4.7	5	4	†.....
Concrete pier.....	3.6	9.0	(d)	(d)	(d)	†(d)

a 27 per cent. settlement.

b 23 per cent. settlement.

c 20¾ per cent. settlement.

d Worthless.

e Gradually cracked to pieces under continuous load equal to 600 feet of rock.

*Free to expand laterally.

†Comparative.

average conditions the settlement would be 5 per cent of the thickness of the bed if ordinary sand were used; an inappreciable amount if granulated slag were used; 10 to 15 per cent with loam, sandy clay, and ashes; and 40 to 60 per cent with dry packing. Under the conditions at Frank the coal pillars left merely serve "to delay the process (of movement) for under the great pressures due to depth, shales, such as here constitute the hanging wall, will 'flow' and seal all openings."*

*Daly, R. A., Miller, W. G., and Rice, G. S. "Report of Commission Appointed to Investigate Turtle Mt., Frank, Alberta, 1911." Can. Dept. Mines, Geol. Survey Branch, Memoir No. 27, p. 30.

TABLE 17.
EXTENT OF FILLING IN RUHR COAL DISTRICT, GERMANY.

Per Cent. of Compression Referred to Original Thickness	Area Worked Out Square Metre	Average Depth from Surface	Age of Workings at Time of Reopening	Composition of Filling
29	14,400	370	8	Waste rock, slates, and sandstones from surface.
28	20,800	450	2	Granulated slag and waste rock (clay and slate).
37	104,000	360	5	Waste from seam and from roof and footwall.
39	26,400	300	4	Waste from seam and from roof and footwall.
60	36,000	270	5	Waste rock from bottom of gang- ways.
21	21,000	380	2	Waste rock from surface, granu- lated slag and clay slate.
28	25,000	440	2	Same as preced- ing.

CONSTRUCTION OVER MINED-OUT AREAS.

When a building is threatened by subsidence resulting from mining operations, or when it is planned to erect a structure upon land which has been undermined and which does not offer sufficiently stable material for a foundation, various steps may be taken to prevent damage to the structure erected or proposed.

Owing to the danger of surface subsidence, the Central Railroad of New Jersey introduced sand into the old mine workings beneath the site of a proposed depot in Scranton in 1911. The Diamond and the Rock seams had been worked and after investigation of the workings it was decided that it would not be necessary to fill the entire area of the workings, but only to reinforce sufficiently the smaller pillars in both seams and fill the wider areas in the Diamond seam so as to prevent any further caving of the roof. In an 8-inch borehole, drilled for this special purpose, a 6-inch pipe was placed. The depth to the lower seam was 80 feet. Sand was brought in railroad cars and flushed into the workings, a total of 9,400 cubic yards being placed at a cost for labor of 29 cents per cubic yard of sand filling.*

*Bunting, D. "Pillar and Artificial Support in Coal Mining." Penn. Legis. Jour., Appendix V, p. 5989. 1913.

The problem of constructing a six-story building, 60 feet wide by 157 feet 7 inches long, on Wyoming Avenue, in Scranton, Pa., was solved by constructing a series of concrete columns. The Big, or 14-foot, bed was close to the surface and had been mined beneath the property, but no maps were available to show the exact location and size of the pillars, and the old workings were inaccessible. Beneath the 14-foot bed other thinner beds had been worked. Five lines of holes were drilled to the rock under the 14-foot bed, the average depth being 40 feet. They were spaced 14 feet 10 inches in one direction and 16 feet 4 inches in the other. Twelve-inch steel pipes were driven into the holes and filled with concrete and, on the top of these, reinforced concrete beams were built.*

The Scranton Electric Company flushed ashes into the old workings under its new power house on Washington avenue. At the present time it is sinking a shaft to be used for dumping ashes into these workings, thus avoiding the expense of hauling them away.

In Pittsburgh, Pennsylvania, the residence section of the city extends over areas from which coal has been mined and it has been thought advisable to construct special foundations under buildings which might be endangered by surface subsidence. Exploratory holes at Beacon Street and Shady Avenue showed that the mine workings were 35 to 55 feet below the surface. Some of the roof had fallen, but some pillars had been left and it was anticipated that subsidence might not be uniform. A pillar of coal extended under one corner of the site for a house. Holes 10 and 14 inches in diameter were drilled to the rock below the coal and six concrete columns were constructed in order to provide support for that part of the house which would be unaffected by caving over the rooms in the mine. No column was constructed under the corner of the house where the coal pillar was located. The concrete columns were 8 inches and 12 inches in diameter inside the galvanized-iron lining which was placed in each hole. The lining was slightly smaller than the hole so that the rock might sink without disturbing the columns. Each column was reinforced and upon these columns were erected reinforced concrete girders which served as a foundation for the house.†

When it is proposed to remove all the mineral in a horizontal bed beneath a structure, it is advisable to mine out the coal in advance in a direction at right angles to the longer axis of the structure and to

*Stevenson, G. E. "Founding a Building Over Coal Mine Workings." Eng. News, Vol. 71, p. 791, 1914.

†"Concrete Column Foundation for a Building Over Coal Mine Workings." Eng. News, Vol. 117, p. 632, 1912.

advance the face at a uniform rate as rapidly as possible so that the structure may be subjected to stress for as short a period as possible.

Reference has previously been made to the special types of construction employed in buildings* and bridges† when surface movement is anticipated. Foundations may be reinforced, long buildings may be divided into units, joints permitting expansion and contraction may be provided, expansion pieces may be placed in railroad tracks, pipe line, cables, etc. In cities in German coal mining districts gutters and curbing are laid with elastic and waterproof joints. Asphalt, cement, and concrete pavements are not used because they are not easily repaired.

RESTORING DAMAGED LANDS.

When subsidence causes breaks and pit holes in agricultural lands, the surface may be rendered temporarily almost valueless for certain kinds of tilling. When the land is of great value for farming, these holes may be filled with waste rock from the mine, cinders, and other refuse, to within four feet of the surface. The remainder of filling necessary to restore a regular surface slope should consist of good soil. At a number of mines in Illinois where such surface damage has resulted from mining operations, the mining companies cooperate with the farmers in filling the pit holes with mine rock.

When subsidence does not break the surface but simply causes shallow basins below the general drainage levels, large ponds form during the spring and may result in the permanent flooding of valuable land. In Northern Illinois, in the longwall field, the topography is such that tile drains have been laid to permit the use of the land. Longwall mining frequently causes a surface movement sufficient to destroy the usefulness of such artificial drainage systems. Referring to the problem in Northern Illinois, G. S. Rice said: "It may be solved to a certain extent through draining the sunken lands by pumping, but even with such a method, aside from the expense, there is a serious difficulty from storm water. When the subsidence is from 2 to 4 feet it will render previously level lands of little use for raising crops until the particular area has come to full settlement and has been retiled. If it were possible to systematize mining so that the land nearest the water courses was first undermined and then in succession the land further away, the damage done to farming would be minimized."‡

*P. 63.

†P. 60.

‡Rice, G. S. "Mining Wastes and Costs in Illinois." Geol. Survey, Bul. No. 14, p. 48.

CHAPTER VII.

LEGAL CONSIDERATIONS.

RIGHT OF SUPPORT.

The title to the minerals, and the right to work them may be held separately from the surface. Under the common law the owner of the surface is entitled to surface support, even though the owner of the minerals finds it impossible to remove them without disturbing the surface. Moreover the owner of the surface is entitled not only to vertical support, but also to lateral support from his neighbors even to the extent that minerals upon adjoining lands cannot be removed in such a manner or to such an extent that the surface of adjoining properties is disturbed.

Leases of coal rights often state distinctly that the lessee shall not be liable for damage to the surface, and where surface rights only are sold, the deed often states that the title to the surface does not include the right to surface support if the owner of the mineral rights mines out the mineral. In spite of such clauses in deeds and in leases suits are of common occurrence when surface and mineral rights are owned by different parties.

MINING UNDER MUNICIPALITIES.

The problem of the claims of municipalities in the coal districts has aroused considerable discussion. In many instances coal mines have been opened upon lands remote from towns and upon which no buildings other than the mine structures were erected at the time. Later mining villages have grown up near the mines and residences and other buildings have been constructed upon the land which had previously been undermined. In many instances, owing to the importance of locating near an abundant fuel supply, industrial plants have been erected in these mining villages or in other towns in the coal district. Eventually large cities have grown up on the lands on which coal mining was the pioneer industry. Similarly mines have been opened outside the limits of important cities and mining operations have been confined to the area which was outside the limits of the city when the mine was opened, but in the course of years the city has extended its limits to include the mine and the area undermined.

The claims of the municipality upon the mining interests, which may have a right by contract and under the law to mine all the coal and to be exempt from liability for damages to the surface, were forcibly presented by Mayor B. Dimmick of Scranton before the Pennsylvania Anthracite Mine Cave Commission, as follows:

"I am of the opinion that there is no constitutional barrier against the inclusion in the general police power of a state or a community of the specific power to declare as null and void and as against sound public morals any and all contracts that waive the right to a reasonable support of the surface which is to be occupied and used for community purposes. I would recommend submitting to the Legislature an act that would declare null and void and as being contrary to public policy any and all contracts that waive the right to, or release from responsibility for, reasonable support of the surface wherever such surface is actually devoted to community life.

"Fortunately this problem has been attacked at a period in which public opinion is slowly but surely crystallizing in favor of acceptance of two general principles, the first being in the direction of such qualifications to the ownership and use of property as are exacted through the increasing interdependence of modern life, such qualifications being in no sense a redistribution of property, in no sense a taking away from one and giving to another, but simply such restrictions and regulations as are demanded, not only in the carrying out of the ancient rule, 'So use your own as not to injure another,' but also for the general welfare of the community. The second principle is that Society must accommodate itself to such costs as are incident not only to fair return to both capital and labor, but also to all the accidents and burdens that result from any activities that Society desires or is compelled to enjoy, and this principle is being regarded as so clearly equitable that its enforcement is being demanded, all private contracts to the contrary notwithstanding. If this principle can and should be enforced when the health of the community or individual is at stake, surely it can and should be enforced when, as in the case of support of the surface, the very lives of men, women, and children are jeopardized.

"The maintenance of the surface, upon which are located the communities that extract the coal, should be regarded as a necessary factor in the cost of mining and should be paid for by the consumer. Such inclusion of the cost of the support of the surface in the general cost of production will be fairer than any fixed tax to be imposed by

the State and then paid out, say to municipalities, to be expended in securing such support.

"It is possible that even under the existing welfare clauses of the acts governing municipalities of Pennsylvania, the proposed exercise of police power might be upheld, but certainly the hands of these municipalities in the anthracite region would be greatly strengthened by such proposed legislation.

"In contemplating this exercise of police power, I realize that there is possibly no exact precedent therefor, yet such exercises would clearly fall within not only the modern but even the ancient definition of the power. So eminent a judge and publicist as Jeremiah S. Black once said that "the police power of the State, of which she cannot disarm herself if she would, enables her to regulate the use even of private property in such manner that neither the general public nor particular individuals can be made to suffer by it unjustly."

With these claims of Mayor Dimmick many eminent lawyers have taken issue.*

Opinions upon a number of the points under discussion are given in the following citations:

"In the natural state of land one part of it receives support from another, upper from lower strata, and soil from adjacent soil." (Per Lord Selborne in *Dalton v. Angus*, 6 A. C. 791.)

"Where the surface belongs to one and the minerals to another, no evidence of title appearing to regulate or qualify their rights of enjoyment, the owner of the minerals cannot remove them without leaving sufficient support to maintain the surface in its natural state." (*Wilms v. Jess*, 94 Ill. 464, 1880.)

The same principles hold between the owners of different minerals lying in separate beds. If one bed lies above another the owner of the lower bed must give support to the upper bed. (*MacSwinney* p. 301.)

A coal mine operating beneath a clay mine is liable for injuries to the upper mine caused by failure to leave sufficient pillars in the coal mine. (*Yandes v. Wright*, 66 Ind. 319, 1879.)

The right of support is not affected by the nature of the strata nor by the difficulty of propping up the surface. (*MacSwinney* p. 292.)

The right of support is wholly independent of the comparative

*For a complete statement of American cases on the various points of discussion between surface and mining rights, see *Lindley on mines*, Title IX, Ch. II and III, 8d Ed., 1914. For British cases, see *MacSwinney*, R. F. "The Law of Mines, Quarries and Minerals." Ch. XIV, 4th Ed., 1912.

values of the substance receiving and the substance giving support. (*Op. Cit.*, p. 292.)

The right of lateral support is an absolute one. The obligation to respect it is in no way affected by the question of negligence. (50 Mo. App. 525.)

“Every owner of land in its natural state has a *prima facie* right to support, lateral as well as vertical; and the adjacent or subjacent owner has no right, *prima facie*, in order to win his minerals, to withdraw such support. The burden, both in pleading and in proof, is upon him who asserts that the position is different from that existing as of common right.” (*Op. Cit.*, p. 299.)

EXEMPTION FROM LIABILITY FOR DAMAGE TO SURFACE.

A conveyance of the right to mine all the underlying minerals implies that in so mining such minerals the surface land shall be sufficiently supported and that so much of such minerals may be mined as can be obtained without injury to the surface. A waiver of the obligation to support the surface must be made by the owner of the surface land by language clear and unequivocal. Such a waiver does not follow a conveyance of all such minerals, nor from the use of language in such a conveyance to the effect that the mining operations shall be “conducted with as little damage to the surface as conveniently may.” (*Seitz v. Coal Valley Mining Co.*, 149 Ill. App. 85, 1909.)

Where a land owner sells the surface, reserving to himself the minerals with power to get them, he must, if he intends to have power to get them in a way which will destroy the surface, frame the reservation in such a way as to show clearly that he is intended to have that power. (*Wilms v. Jess*, 94 Ill. 464, 1880.)

When an instrument excludes the right of the surface owner to support, the mine owner may be liable, if he works negligently, or contrary to the custom of the country. (*MacSwinney* p. 311.)

The right of support by land in its natural state may also be excluded, wholly or in fact, by statute. Examples of this may be found in various English Acts. (See *MacSwinney* p. 312.)

In an investigation of the surface damage in a section of Scranton, Pennsylvania, it was found that 48 per cent of the titles contained a clause completely waiving surface support, in the following language: “All the coal in, under and upon said lot, together with the sole right and privilege to mine and remove all the coal under said lots without

incurring in any event whatever any liability for injury or damage done to the surface of said lots or improvements thereon or that may thereafter be put thereon caused by mining or removal of said coal."

Fourteen per cent of the titles contained waivers which are more or less conditional in their nature: "All the anthracite coal lying underneath, also half the width of streets adjoining. It being understood and agreed that at least one-fourth thereof, properly distributed, shall be left for surface support and the coal shall be mined in a workman-like and skillful manner, it being understood that all the coal is to be mined and paid for except so much left thereof as may be necessary to be left for pillars to support the surface thereof, and it being possible that there may be a difference of opinion relating to the fulfillment of this provision it is agreed that the matter shall be submitted to a board of competent and skillful engineers, each party to select one and, in case of failure to agree, said engineers are empowered to call in a third mining engineer and the decision of the majority shall be final."

Ten per cent of the titles contained the following clause: "All the coal and minerals under said lot, together with the right to mine and remove all of said coal and minerals, provided also that in removing the coal the second party shall leave one-fourth thereof in place for the protection of the surface."

The remainder of the titles examined by the investigators contained the following clause: "All right, title, etc., to all coal in and under said lots, also the coal under the surface in front of said lots to the center of the street."

In the report of the Pennsylvania State Anthracite Mine Cave Commission excerpts of 42 deeds are given showing the various forms in which reservations have been made when the title to the surface has been severed from the mining right.*

In the bituminous fields a customary form of exemption clause in deed for coal, separate from the surface, is as follows: "All the coal underlying and within the described lands together with the right to take the entire quantity, or a less quantity of said coal, without leaving any support for the overlying strata, and without liability for any injury or damage which may result from the breaking of said strata." Another type of exemption clause employed in Illinois is as follows: "Releasing and surrendering any and all claims for dam-

*Pa. Legislative Journal, Appendix, Vol. 5, p. 5953, 1913.

ages and all liability by reason of damages either to person or property which may in any way be caused or occasioned at any time hereafter, directly or indirectly, by the mining or removing of coal or other minerals.”

PROTECTION OF SURFACE BY GRANTS AND BY LEGISLATION.

The right of support for land in its non-natural state may be acquired by express or implied grant.* Where land is severed from adjoining land for a particular purpose, and such purpose is known at the time the mineral right is severed from the surface, there is, *prima facie*, an implied grant of a reasonable degree of support for carrying out the particular purpose under consideration.

In order to protect railways, canals, waterworks, sewers, etc., the British Parliament has enacted legislation which guarantees that such structures shall not be undermined, if liable to be damaged, without notice. In accordance with these acts, the mining company may be required to leave a pillar, but the mining company is compensated for the coal left in the ground and for any damage that may be sustained by the interruption with the system of mining. In the case of the Railway Act, it is specified that “the (railway) company shall from time to time pay to the owner, lessee, or occupier of any such mines, extending so as to lie on both sides of the railway, all such additional expenses and losses as shall be incurred by such owner, lessee, or occupier by reason of the severance of the lands lying over such mines by the railway, or of the continuous working of such mines being interrupted, or by reason of the same being worked in such a manner and under such restrictions as not to prejudice or injure the railway, and for any minerals not purchased by the company which cannot be obtained by reason of making and maintaining the railway; and if any dispute or question shall arise between the company and such owner, lessee, or occupier as aforesaid, touching the amount of such losses or expenses, the same shall be settled by arbitration.†

Similar sections are included in the Wat. Cl. Const. Act, 1847; the Pub. Health Act, 1875 (Support of Sewers); Amendment Act, 1883; the Local Government Act, 1894; the Small Holdings and Allotments Act, 1908; and the Housing and Town Planning Act, 1909.‡

Under these acts, as noted, the intention of a mining company to

*MacSwinney Pp. 315, 316.

†Railway Clauses Consolidation Act, 1845, Sec. 81. 8 and 9 Vict., c. 20.

‡MacSwinney P. 370.

remove the mineral beneath any of the legally specified structure must be announced through a regular "notice of intention to work." This notice is given in most cases thirty days in advance of the intended working. If, after notice has been given, the owner of the structure does not agree to negotiate with the mining company, it is lawful for the mining company to proceed in the regular manner of working. If damage results, it shall be repaired by the mining company.*

Gas works and gas mains are not within the British mining code excepting in so far as they are vested in local authority. Private gas companies are not entitled to support if the mineral right has been severed, but the colliery company would be liable for damages to gas pipes and leakage of gas. In the absence of special provisions, owners, lessees, and occupiers of mines are not liable for damage caused to tramways by working mines or minerals in the usual and ordinary course.† This mining code of Great Britain is not applicable to burial grounds, school sites, public highways, bridges, nor canals.

In several states of the United States there are statutes in regard to support, particularly in western states in which the lode mining law permits extralateral mining. The statutes of Colorado, for example, prescribe that "when the right to mine is in any case separate from the ownership or right of occupancy to the surface, the owner or rightful occupant of the surface may demand satisfactory security from the miner, and if it be refused, may enjoin such miner from working until such security is given." No person shall have the right to mine under any building or other improvement unless he shall first secure the parties owning the same against all damages, except by priority of right."‡

Other states, such as Idaho, North Dakota, South Dakota, and Wyoming, have similar laws. In commenting on this type of legislation, Lindley says, "We are not aware that this class of legislation has been the subject of judicial investigation. It seems to us that such legislation is not altogether free from constitutional objections."§

Arkansas has a law, approved Feb. 28, 1907, forbidding the mining of coal or any other mineral substance from beneath a cemetery or burial place. No openings whatever may be driven under or through

*Cockburn, J. H. "Minerals Under Railways and Statutory Works." Trans. Inst. Min. Engrs., Vol. 39, p. 104, 1909.

†Op. cit. P. 128.

‡Rev. Stat., Colorado, p. 4213, 1908.

§Lindley "American Law Relating to Mines and Mineral Lands." Vol. 3, p. 2016. 3d Ed.

the mineral directly beneath the cemetery under penalty of a fine of \$5,000 or imprisonment of from one to five years.*

In Pennsylvania the Davis Mine Cave Act, approved July 26, 1913, provides regulations governing the mining of coal and other minerals and the "support underlying and beneath the surface of the several streets, avenues, thoroughfares, courts, alleys, places, and public highways within the limits of the several municipal corporations, and authorizing the creation of a Bureau of Mine Inspection and Surface Support," by any municipal corporation within the anthracite coal fields. Members of the Bureau of Mine Inspection and Surface Support have the right and power to enter, examine, and survey any mine within the limits of the municipality. Mining companies are required to furnish accurate and complete maps of the workings and to keep the same up to date.

The mining companies are required to "maintain, uphold, and preserve the stability of the surface" of the various streets, etc. The officers of mining companies are made responsible for the violation of the provisions of the act and for violation are subject to a fine of \$1,000 or imprisonment for ninety days, or both.†

An ordinance was enacted by the Borough of Plymouth, Luzerne County, Pennsylvania, forbidding mining within 200 feet of the street lines and as the borough is platted in 400-foot squares, this prohibited any mining whatever. The county courts in Borough of Plymouth v. Plymouth Coal Co. restrained the coal company from mining under the streets.

REMEDIES.

In the event that the owner of the surface is entitled to surface support and is sustaining damages by the mining operations beneath or adjacent to his land he may recover damages or if the damage is irreparable or immeasurable he may apply for an injunction to restrain mining operations. If the mining operations are being conducted by parties whose financial resources are not adequate to insure the payment of damages in case such are assessed, an injunction may be issued.

The right of support is not infringed by excavation, but by subsidence and damages do not exist until subsidence has actually occurred. (Catlin Coal Co., v. Henry Lloyd, 109 Ill. App. Rep. 122, 1902.) The Pennsylvania court now holds, however, that the cause of action

*Arkansas Acts of 1907, Sec. 566 d-f.

†Pa. Acts of General Assembly, 1913, No. 857.

accrues when the support is removed and is barred after the lapse of six years from such removal. It is said by the Pennsylvania court that the adoption of any more onerous rule "would encourage the purchase of surface over coal mines for speculation in future law suits." (Noonan v. Pardee, 200 Pa. 474, 86 Am. St. Rep. 722, 55 L. R. A. 410.)

"The owner of the subsidence estate is not liable to the surface proprietor for a subsidence caused by excavations made by his predecessor in title, although damage does not occur until after such owner came into possession. This results from the fact that, while the subsidence gives the cause of action, the responsibility therefor attaches to him whose acts and omissions have brought about the mischief."*

If the owner of real estate which has been injuriously affected or damaged by a permanent structure has not brought an action to recover damages and conveys the land to another, the cause of action does not pass with the title nor inure to the benefit of the depreciation in value in the price paid. (La Salle County Carbon Coal Co. v. Sanitary District of Chicago, 260 Ill. 423, 1913.)

Depreciation in the value of the surface caused by the mere apprehension of future damage gives no cause of action.† Only damage which has actually occurred may be considered by a court, but each fresh subsidence constitutes a basis for a new claim for damages. (Catlin Coal Co. v. Henry Lloyd, 124 Ill. App. 394, 1906.)

"The right of support is not infringed unless the subsidence is substantial. There must be some real sensible interference with the land. The right of support in ordinary cases is infringed where the subsidence is substantial, but the damage is inappreciable; and it is now settled that an injunction may be obtained where the subsidence is substantial, although the damage is inappreciable. The right of support is analogous to a right of property, and is a right to have the surface kept securely at its ancient and natural level."‡

A mine owner cannot avoid liability by showing that his workings have been proper and in the customary manner. "The act of removing all support from the superimposed soil is, *prima facie*, the cause of its subsequently subsiding." (Wilms v. Jess, 94 Ill. 464, 1880.)

Where land has been artificially burdened by a building and no contract or prescription is available to regulate its right to support, no right to support, lateral or vertical, exists for the building. In an

*Lindley P. 2021.

†MacSwinney P. 294.

‡MacSwinney P. 297.

action for removing support from land artificially burdened, the plaintiff has always been obliged, as a matter of pleading, to show that he is entitled to have the weight supported.*

Where the injury to the surface would have resulted from mining operations if no buildings existed upon the surface, the act creating the subsidence is wrongful and renders the owners of the mine liable for all damages that result from mining to the buildings as well as to the land itself. (*Wilms v. Jess*, 94 Ill. 464, 1880.)

As a general rule, the measure of damages in actions for injuries to real property is the difference in market value before and after the injury to the premises. But to this rule there are exceptions, and it has been held that the cost of repair or of restoring the premises to their original condition is the true and better rule to apply. The valuation should be adopted which will be most beneficial to the injured party, for he is entitled to the benefit of the premises intact. (*Donk Bros. Coal and Coke Co. v. Slata*, 133 Ill. App. 280; 135 Ill. App. 633.)

A bill in equity to restrain the mining of coal was dismissed as there is a remedy at law in case damage is done to surface by subsidence. (*Henry Lloyd v. Catlin Coal Co.*, 109 Ill. App. 37, 1902.)

What amount of coal may be safely mined and what amount must be left for necessary support of the soil are largely engineering questions, and it is only in rare cases, where the remedy at law is so inadequate as to render such course necessary, that a court of equity will direct the work by injunction. (*Henry Lloyd v. Catlin Coal Co.*, 210 Ill. 460, 1904.)

As previously noted† considerable damage has resulted in England from the pumping of brine. Under the Brine Pumping Act of 1891 (54 and 55 Vict. c. 40), upon application, compensation districts may be formed within the pumping fields. For every compensation district, under the act, there is established a compensation board. This board is incorporated and consists of representatives of the various interests concerned; one-third shall be persons not interested in the brine business and appointed by the county council; one-third elected by the brine pumpers; and one-third (not interested in brine pumping) appointed by the local sanitary authority. A compensation fund is maintained in each district; upon each pumper is levied a tax not exceeding 3 pence per 1,000 gallons pumped for a twelve-month period. Out of this fund damages allowed by the compensation board are paid.

*MacSwinney P. 304.

†Ch. I.

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UNIVERSITY OF ILLINOIS BULLETIN

ISSUED WEEKLY

VOL. XIV

SEPTEMBER 18, 1916

NO. 3

[Entered as second-class matter Dec. 11, 1912, at Urbana, Ill., under the Act of August 24, 1912.]

THE TRACTIVE RESISTANCE ON CURVES OF A 28-TON ELECTRIC CAR

BY
EDWARD C. SCHMIDT
AND
HAROLD H. DUNN



BULLETIN NO. 92
ENGINEERING EXPERIMENT STATION

PUBLISHED BY THE UNIVERSITY OF ILLINOIS, URBANA

PRICE: TWENTY-FIVE CENTS
EUROPEAN AGENT
CHAPMAN AND HALL, LTD., LONDON

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UNIVERSITY OF ILLINOIS

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THE TRACTIVE RESISTANCE ON CURVES OF A 28-TON ELECTRIC CAR

I. INTRODUCTION

The tractive resistance of cars running on curved track is greater than their resistance on straight track of like grade and construction. This excess is generally termed curve resistance or resistance due to curvature. A knowledge of its magnitude is needed in many problems which present themselves in connection with steam and electric railway design and operation. While these problems are more important and more numerous on steam roads, they are not unimportant nor infrequent on electric roads. Nearly all the existing information regarding curve resistance has arisen from tests and experience with trains on steam railways, and it is doubtful whether the values of curve resistance thus derived are valid for the single self-propelled cars used on electric lines. Such considerations led the Railway Engineering Department of the University of Illinois to make the tests, the results of which are presented in this bulletin.

The tests were undertaken to measure the curve resistance of a 28-ton electric car, owned by the department, with the view of determining its value at various speeds and on as great a variety of curves as were available on the lines of the Illinois Traction System, upon which the experiments were conducted. The results established for this car the relations between curve resistance and speed and between curve resistance and rate of curvature.

The test conditions, test methods, and final results are presented in the body of the bulletin, while the details concerning the apparatus, test data, methods of calculation, and intermediate results are given in the appendixes. Throughout the bulletin the term "curve resistance" or "resistance due to track curvature" means the tractive force needed by the car on curves in excess of the force needed to move it over straight track. This tractive force is the force required at the wheel rims to keep the car moving at uniform speed on level track and in still air. It is expressed in pounds per ton of car weight.

Acknowledgment is gratefully made of the interest and coöperation of the officers of The Illinois Traction System, which rendered it possible to conduct the tests on that road; and to Mr. D. C. Faber, formerly a Fellow in the Department of Railway Engineering, who was in charge of the test car during some of the tests and who made some of the preliminary calculations.

II. SUMMARY

At the expense of some duplication there is presented at this point a summarized statement of the conditions, methods, and results of the tests, which may serve to provide a general view of the work and to facilitate an understanding of the more detailed explanations which follow.

1. *The Car, Track, and Equipment.*—The tests were made with a car such as is commonly used on interurban electric roads; it has a body 45 feet long of the double-end type with round vestibules. This body is carried on four-wheeled trucks which are spaced 23 feet 3 inches from center to center and which have a wheel base of 6 feet 4 inches. The car is equipped with four 50-horsepower motors and weighs approximately 28 tons.

The tractive resistance of this car was determined when running upon each of seven curves whose curvature varied from 2 to $14\frac{1}{2}$ degrees, and also when running upon adjacent tangent track. The superelevation of the outer rail on the curves varied from 0.75 inches on the 2-degree curve to 5.9 inches on the $14\frac{1}{2}$ -degree curve. The track was laid with 70-pound rails on ties spaced about 24 inches between centers in gravel or cinder ballast. Judged by the standards which prevail on electric interurban roads built for moderate speed, the track was well constructed and well maintained. It was surveyed especially for the purposes of the tests. With a few exceptions the tests were made on dry rail in fair weather. The average air temperature varied during the tests from 25 to 65 degrees F. and the maximum average wind velocity was 18 miles per hour.

2. *Methods.*—In making a test, the test car was run first in one direction and then in the other over one of the curves and its adjacent tangent. During each such pair of runs the car speed was maintained as nearly constant as possible. Similar pairs of runs were then made at other speeds, until sufficient data had been accumulated to define the resistance at various speeds on the curve and on its corresponding tangent. Under this procedure each pair of runs results in two values of resistance: one with such wind as prevailed helping the car, the other with the wind opposing it. From the final curves which define the mean between these values, the influence of the wind is, therefore, nearly, if not quite, eliminated, and the results relate to movement in still air. The results of the tests provide mean values of resistance on each curve and on its tangent. The difference between these values of resistance is the desired resistance due to track curvature.

3. *Results.*—The tests demonstrate that for the car in question curve resistance varies directly with both track curvature and speed. For a particular speed, the curve resistance increases as the curvature increases and in direct ratio with the curvature, as shown in Fig. 16. This implies that at a particular speed the curve resistance, when expressed in pounds per ton per degree of curve, is a constant for all curvatures; a relation which is in accord with the results of previous experiments. On the other hand, the value of curve resistance expressed in pounds per ton per degree is shown to be different at each different speed, and it varies in such a way that for a curve of a particular curvature the curve resistance increases in direct ratio with the speed, as shown in Fig. 17. The concurrent relations between curve resistance, track curvature, and speed are shown in Fig. 18, and they are defined by the formula:

$$R_c = 0.058 S C,$$

in which R_c is the curve resistance expressed in pounds per ton, S is the speed in miles per hour, and C is the degree of curve. Values derived by means of this equation appear in Table 4.

III. MEANS EMPLOYED IN CONDUCTING THE TESTS

4. *The Test Car.*—The car used for the tests is owned by the Railway Engineering Department of the University. It is a standard 45-foot car similar to those commonly used on interurban roads built for moderate speed. Its general design is shown in Figs. 1 and 2.



FIG. 1. THE TEST CAR

The car weighs 55,150 pounds, although during the tests this weight was subject to certain corrections due to changes in equipment and in the number of passengers. The sectional area of the car body and trucks is 90 square feet.

The trucks are of the standard Motor Truck Company's C-60 type and weigh 7,824 pounds each, without the motors. The truck journals are $4\frac{1}{4}$ by 8 inches, the wheel diameter is 33 inches, and the

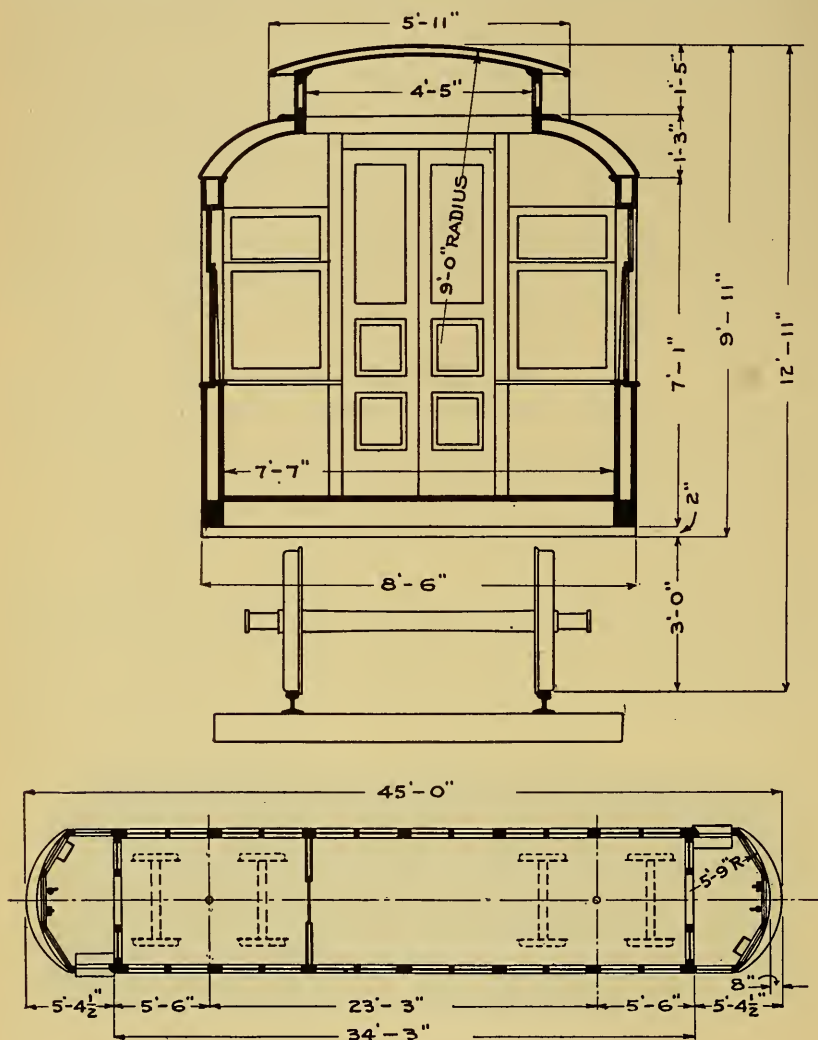


FIG. 2. PLAN AND CROSS SECTION OF THE TEST CAR

truck wheel base is 6 feet 4 inches. One of the trucks is equipped with rolled steel wheels and the other with chilled cast iron wheels, all of which have standard Master Car Builders' tread and flange contours. During these tests the car was equipped with ball-bearing center

plates. Each truck is provided with two Westinghouse 101-D, 500-volt, direct current motors, which have a commercial rating of 50 horsepower each. The motors are mounted on the axles and geared to them in the ratio of 22 to 62. They are controlled by the Westinghouse unit switch system of multiple control.

An especially designed recording apparatus within the car offers a means for measuring and recording the current consumed, the voltage, speed, time, distance traversed, location on the road, and brake cylinder pressure. Continuous graphical records of these data are drawn upon a chart which is made to travel at a rate proportional either to time or to the distance traveled by the car. A more complete description of the car and of this recording apparatus appears in Bulletin 74 of the University of Illinois Engineering Experiment Station. In addition to the data above enumerated, there were recorded for each run the average wind velocity and direction, air temperature, rail condition, and the gross car weight.

5. *The Track.*—The tests were made on the lines of the Illinois Traction System between Danville, Urbana, Champaign, Decatur, and Springfield. The oldest portions of this track were laid in 1903, the latest in 1907. Judged by the standards which prevail on interurban electric roads built for moderate speed, the track was well constructed and well maintained. On both the curves and the tangents used in the tests, the track was laid with 70-pound A. S. C. E. section rails carried on hard-wood ties spaced about 24 inches between centers. The ballast was either gravel or cinders.

The curves chosen were seven in number varying in curvature from 2 to $14\frac{1}{2}$ degrees, as great a range in curvature as was presented by the track available for test purposes. The gauge on all but the 8-degree and the $14\frac{1}{2}$ -degree curves was 4 feet $8\frac{1}{2}$ inches, while on these two curves it was 4 feet 9 inches. A summary of the facts relating to the seven curves is given in Table 1. All the track used for

TABLE 1
DATA RELATING TO THE SEVEN CURVES

Average Curvature	Length of Track Section	Superelevation			Grade	Weight of Rail
		Average	Maximum	Minimum	Difference in Elevation	
Degrees	Feet	Inches	Inches	Inches	Feet	Lb. per Yd.
2°-0'	500	0.7	1.2	0.0	1.47	70
2°-50'	1737	3.0	4.1	1.3	8.52	70
3°-40'	1125	1.9	3.3	0.2	4.81	70
5°-0'	714	2.8	3.8	0.4	0.15	70
6°-30'	365	4.5	5.4	2.2	3.22	70
8°-0'	176	5.3	5.5	5.0	0.65	70
14°-30'	276	5.9	6.9	5.0	0.17	70

the tests was surveyed especially for the test purposes, and the results of these surveys are presented in Appendix I, together with further details.

IV. TEST CONDITIONS AND TEST METHODS

6. *Test Conditions.*—The tests were all made in moderate weather. The lowest air temperature recorded during any test was 15 degrees F., and the lowest average temperature throughout the duration of any test was 25 degrees. The corresponding highest temperatures were 70 degrees and 65 degrees, respectively. With five exceptions the tests were made on dry rails. The highest average velocity of the wind prevailing during any of the tests was 18 miles per hour, whereas the maximum component of this velocity parallel

TABLE 2
A SUMMARY OF TEST CONDITIONS ON THE CURVES AND ON THEIR
CORRESPONDING TANGENTS

1	2	3	4	5	6	7
Curve or Tangent	Test Number	Degree of Curve	Weight of Car and Load	Approx. Average Air Temp.	Rail Condition	Average Wind Velocity
			Pounds	Deg. F.		M.P.H.
Curve	117-118	2°-0'	56200	25	Wet	12.0
	123-124	2°-0'	56750	30	Dry	4.0
Tangent W	117-118		56200	25	Wet	12.0
	123-124		56750	30	Dry	4.0
Curve	119-120	2°-50'	56750	45	Dry	3.5
	121-122	2°-50'	56750	30	Dry	3.0
Tangent S	119-120		56750	45	Dry	3.5
	121-122		56750	30	Dry	3.0
Curve	125-126	3°-40'	57350	35	Dry	15.0
	127-128	3°-40'	57500	25	Wet	0.0
	129-130	3°-40'	57350	65	Dry	15.0
Tangent R	125-126		57350	35	Dry	15.0
	127-128		57500	25	Wet	0.0
	129-130		57350	65	Dry	15.0
	141-142		57800	40	Dry	3.8
	153-154		57900	60	Dry	0.0
Curve	125-126	5°-0'	57350	35	Dry	15.0
	127-128	5°-0'	57500	25	Wet	0.0
	129-130	5°-0'	57350	65	Dry	15.0
Tangent R	*					
Curve	133-134	6°-30'	57300	35	Wet	7.5
	109-110		56950	55	Wet	18.0
Tangent D	111-112		56350	55	Wet	18.0
	113-114		56200	55	Dry	10.0
Curve	141-142	8°-0'	57800	40	Dry	3.8
	153-154	8°-0'	57900	60	Dry	0.0
Tangent R	*					
Curve	141-142	14°-30'	57800	40	Dry	3.8
	153-154	14°-30'	57900	60	Dry	0.0
Tangent R	*					

* Same tangent as for 3°-40' Curve

to the tangent track or to a chord connecting the ends of the curve was 15 miles per hour. The wind velocity and direction were obtained by means of a portable wind vane and an anemometer set up beside the test track. A summary of the conditions for each test is given in Table 2.

7. *The Selection of the Track.*—The primary consideration in the selection of the curves was to have them include as great a variety and range in degree of curvature as the circumstances would permit. When available, those curves were chosen which had at either end a long and comparatively level stretch of straight track, in order that the resistance on the curve and on the tangent might be measured simultaneously and, therefore, under like conditions of wind and weather. In three out of the seven cases, however, the straight track adjacent to the curve was, by reason of its construction or maintenance, or by virtue of the operating conditions which there prevailed, unsuitable for the purposes of the tests, and in these instances the tangent track was chosen elsewhere. In such cases the determination of tangent resistance was made as far as possible under conditions similar to those which obtained during the tests on the related curve. The curves chosen were fairly regular in curvature, and all the track on both the curves and the tangent sections was well ballasted.

8. *General Methods.*—By means of the data which were recorded during each run and which have been enumerated in Chapter III, it is possible to calculate the gross resistance offered to the motion of the car over a given track section. This gross resistance is composed of the resistance due to grade, that due to acceleration, that due to wind,* the net resistance on level straight track at uniform speed, and, on the curved track, the resistance due to curvature. The purposes of the tests require the elimination of the first three of these five elements of gross resistance. The grade and acceleration resistances may be easily eliminated by calculation, but the wind resistance is neither controllable nor to be eliminated by calculation from the data at hand. The elimination of wind resistance may, however, be accomplished by the method of making the tests, and with this end in view the following method of test was used.

Over each track section the car was run in one direction at a predetermined speed which was maintained as nearly uniform as possible. It was then immediately run over the same section at the same speed, but in the reverse direction. Where curved and tangent track

*Throughout the bulletin wind resistance is distinguished from the resistance due to still air, the latter being an inseparable part of inherent or net resistance.

were adjacent they were both included in each of these runs. This process was repeated at various speeds until enough data had been accumulated to define for this track section the relation between resistance and speed up to the maximum speed of operation. The whole group of these pairs of runs constitutes what is designated in the report as a test.

The wind in one case opposes and in the other helps the motion of the car, and each pair of such runs results, when grade and acceleration resistance have been eliminated, in two values of resistance; one value equal to net resistance* plus such resistance as was offered by the wind, the other equal to net resistance minus the resistance offered by the wind. In runs in which the direction of the wind was parallel to the direction of motion of the car, a properly determined mean between these two values is the net resistance itself with the influence of wind entirely eliminated. In runs in which the direction of the wind made an angle with the direction of motion of the car, only that portion of wind resistance due to the component of wind velocity parallel to the track would disappear from the mean value of resistance, and there would remain embodied in this mean† a certain resistance due to the increased wheel-flange friction caused by that component of wind velocity which is normal to the direction of motion of the car. All the resistance curves resulting directly from the tests represent the mean between paired values determined in the way just described. The curves themselves, therefore, define values of resistance in which there remains only that effect of the wind produced by its velocity component normal to the track; that is, they are in error by only a slight excess in flange friction. Under the low wind velocities which prevailed this error is small, and doubtless much less than the casual variations which occur in inherent resistance itself. These conclusions apply to the immediate results of the tests; that is, to the resistance-speed curves shown in Figs. 3 to 13, inclusive. For reasons stated in the next paragraph the error referred to does not, however, appear in the final results.

The simultaneous operation of the car over the curve and its adjacent tangent ensures that the resistances on the curve and on the tangent were obtained under practically identical wind conditions. Whatever error or excess in mean resistance arises from the normal component of wind velocity is, consequently, likewise identical on the

*As used in this connection the term net resistance is assumed to include on the curves the resistance due to curvature.

†The method of determining this mean is explained in Appendix II.

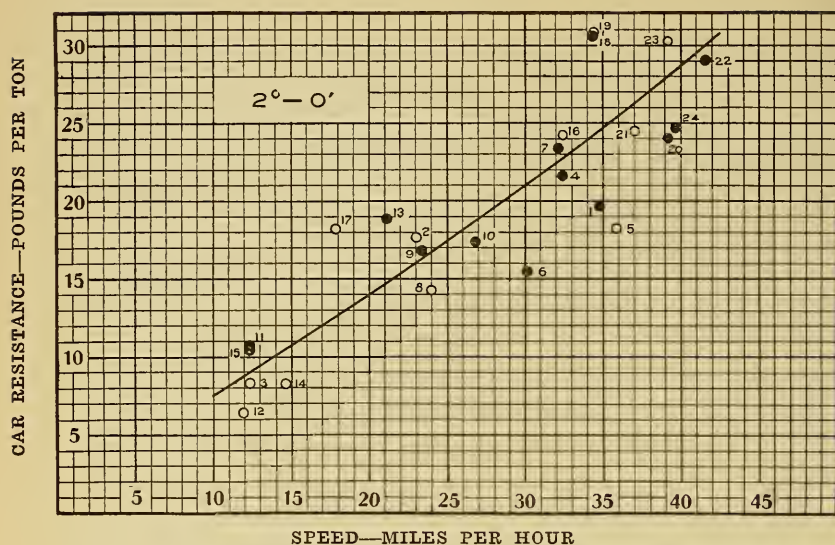
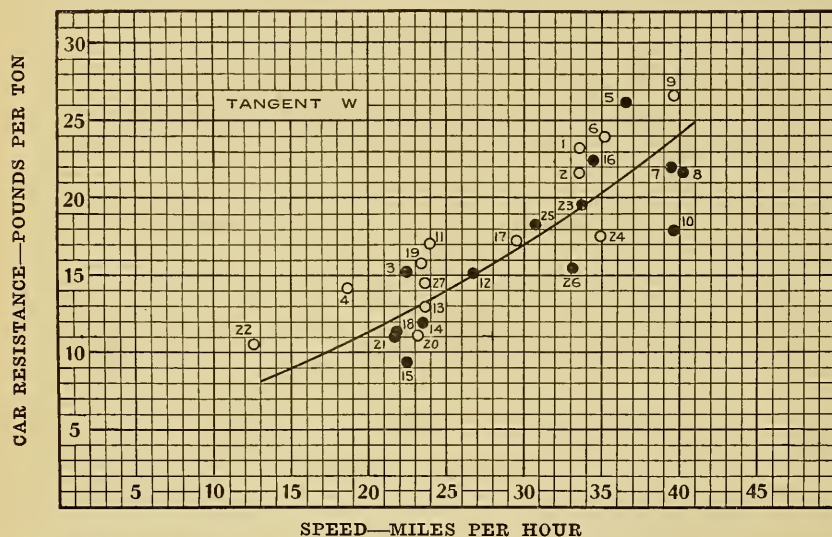
curve and on the tangent. Since, however, the final result sought, the resistance due to curvature, is the difference between the total resistance on the curve and the resistance on the tangent, this error, by the process of subtraction, disappears from the final results of the tests. In those cases in which tangent resistance had to be measured on track not adjacent to the corresponding curve, the wind conditions were nearly the same as during tests on the curve, and the same conclusion, therefore, holds in these instances also.

From the data recorded in the test car the gross resistance was first determined for each run over each track section. This was then corrected for acceleration by means of data provided by the car records, and next for grade resistance by reference to the track profiles. The methods of making these calculations are stated in Appendix II, together with further details concerning the car records and the test methods.

V. THE IMMEDIATE RESULTS OF THE TESTS

The calculations made for each run result, for the track section under consideration, in a value of car resistance at a particular speed. For all the runs these resistance values, together with the corresponding speeds, are set forth in the tables given in Appendix III. Using these values as coördinates, the points shown in Figs. 3 to 13 have been plotted; each point defines the resistance-speed relation during one run, and the whole group of points in each figure characterizes this relation for a particular track section. By a method which is explained in Appendix II, a line has been drawn among the points of each of these figures which represents the mean resistance discussed in Chapter IV, and which in each case has been accepted as defining the relation existing between resistance and speed for the track section in question. Figs. 3 to 13, inclusive, constitute the immediate results of all the tests.

The points indicated by circles in these figures pertain to runs in which the wind opposed the motion of the car, while those represented as black dots pertain to runs in which the wind helped the car motion. The few cases in which there was no wind are represented by circles half filled in. Under the test procedure previously outlined, there should appear in each figure equal numbers of circles and dots. This condition, however, is not exactly met in any of these figures, because during the process of calculation a few points were rejected on account of discrepancies or inadequacy in the record. In general, however, the balance implied by the test procedure is substantially realized.

FIG. 3. THE RELATION OF RESISTANCE TO SPEED ON THE $2^{\circ}-0'$ CURVEFIG. 4. THE RELATION OF RESISTANCE TO SPEED ON TANGENT W, WHICH PERTAINS TO THE $2^{\circ}-0'$ CURVE

There is considerable variation among the points in these figures and also a considerable variation from the mean defined by the curves. Some of this is due to the influence of wind, but even among the points

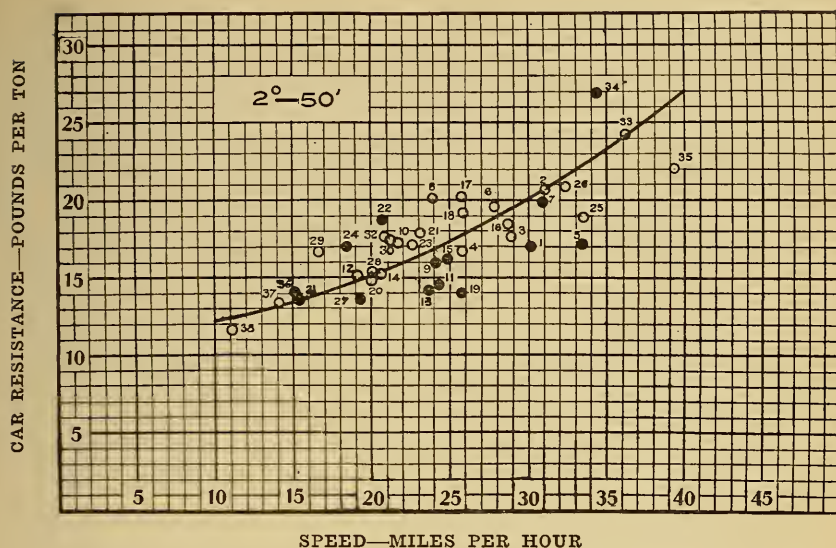


FIG. 5. THE RELATION OF RESISTANCE TO SPEED ON THE 2°-50' CURVE

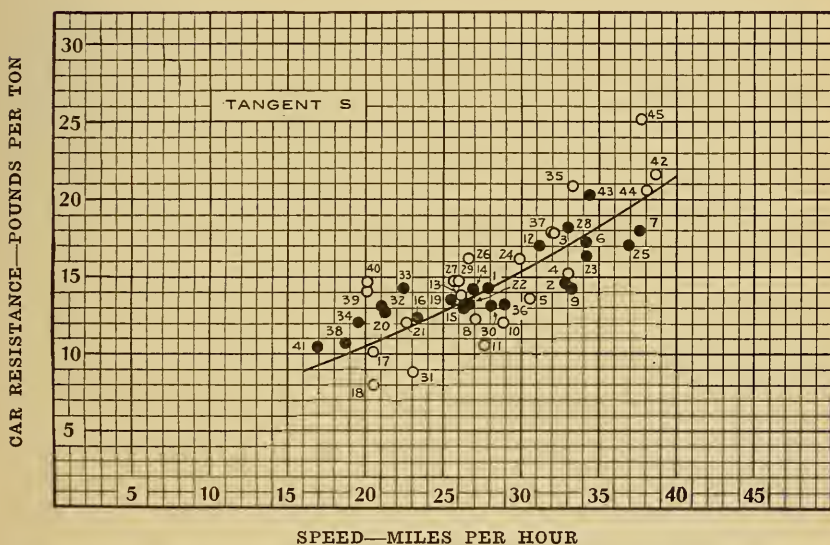


FIG. 6. THE RELATION OF RESISTANCE TO SPEED ON TANGENT S, WHICH PERTAINS TO THE 2°-50' CURVE

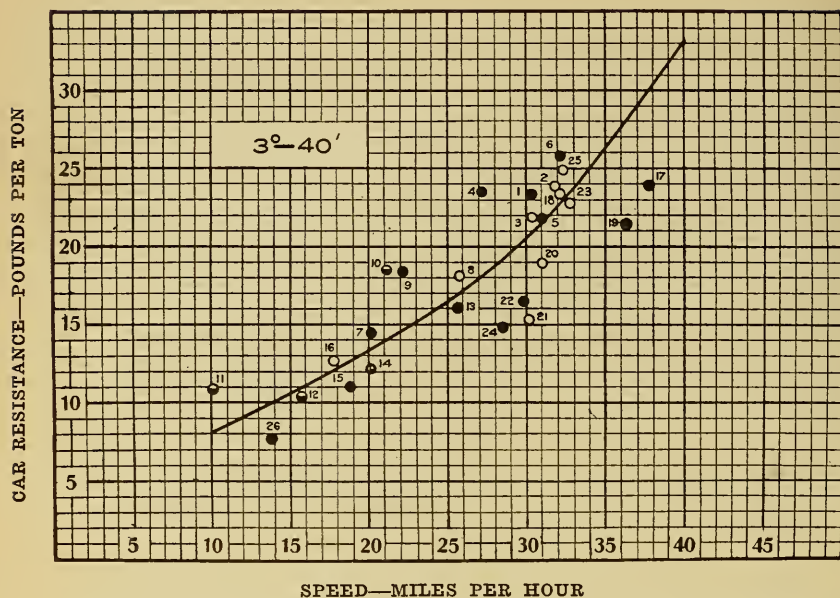


FIG. 7. THE RELATION OF RESISTANCE TO SPEED ON THE 3°-40' CURVE

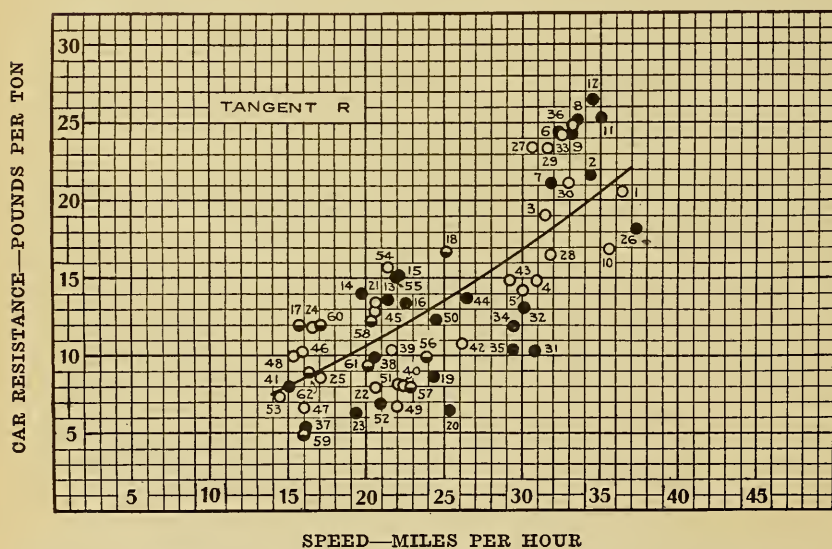


FIG. 8. THE RELATION OF RESISTANCE TO SPEED ON TANGENT R. WHICH PERTAINS TO THE 3°-40', THE 5°-0', THE 8°-0', AND THE 14°-30' CURVES

relating to like wind conditions there is similar disagreement. Part of this variation may be due to accumulated errors in instruments or in the calculations; but, since every precaution was taken to avoid such errors, they are undoubtedly rare and small in amount. The reasons for the differences referred to must be sought chiefly in the casual changes which occur in such elements of inherent resistance as flange friction, journal friction, and gear friction, and perhaps also in instantaneous changes in wind velocity and direction. The discordance shown is not greater than that usually encountered in measurements of car or train resistance.

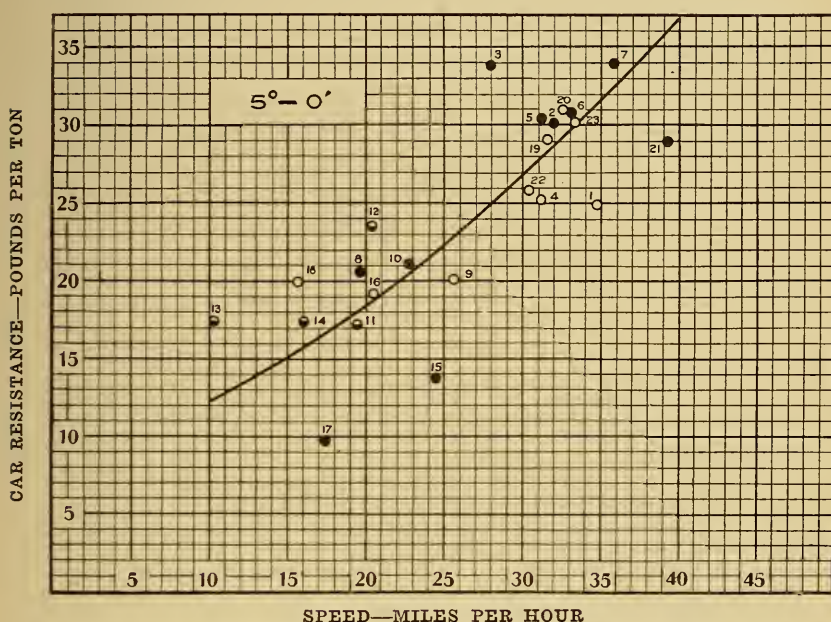


FIG. 9. THE RELATION OF RESISTANCE TO SPEED ON THE 5°-0' CURVE

All the figures show a continuous increase of resistance with speed, except Fig. 13, applying to the 14°-30' curve. In this case the resistance decreases until a speed of about 12½ miles per hour is reached and then increases as the speed is increased beyond this point. This peculiarity, which showed itself during the earliest runs on this curve, led to repetitions of the tests, but always with the same result, and Fig. 13 must be accepted as representing the facts in the case. The superelevation on this curve is 5.9 inches, and the component of car weight parallel to the plane of the rails amounts to about 5,800 pounds. It was assumed that perhaps this component, up to the

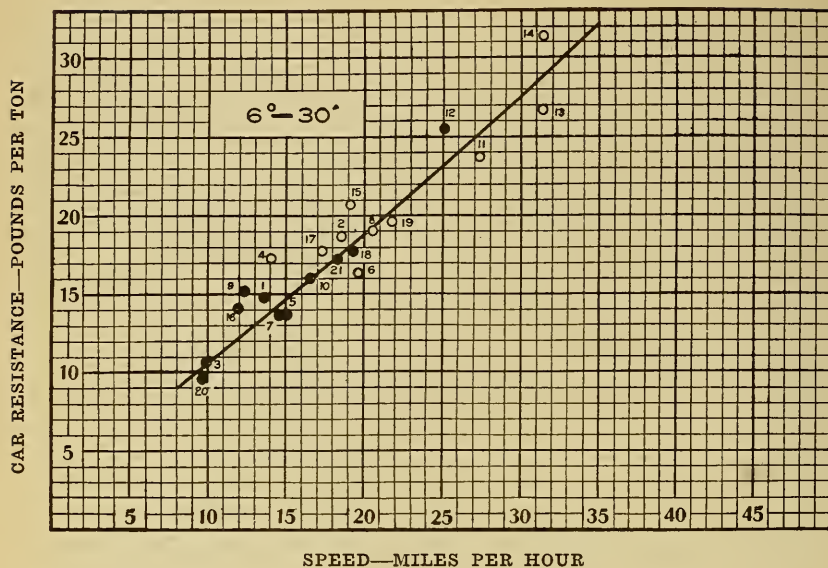


FIG. 10. THE RELATION OF RESISTANCE TO SPEED ON THE 6°-30' CURVE

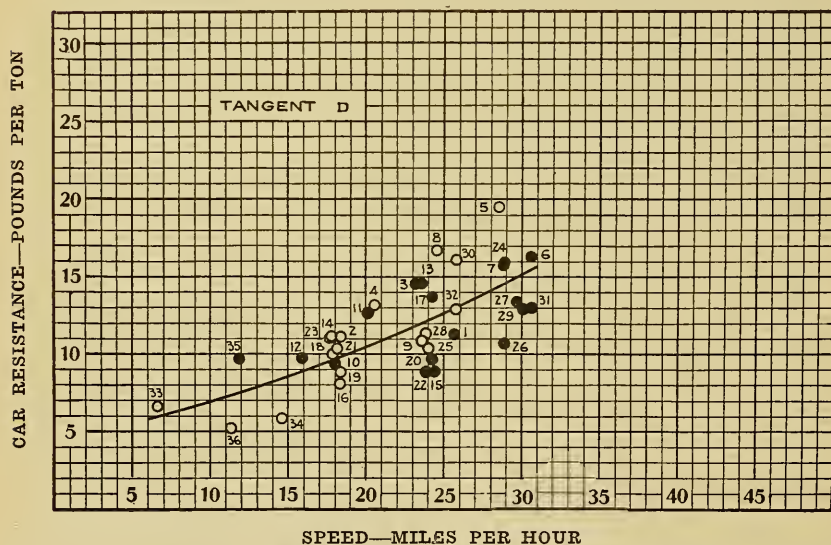


FIG. 11. THE RELATION OF RESISTANCE TO SPEED ON TANGENT D, WHICH PERTAINS TO THE 6°-30' CURVE

speed of minimum resistance, had produced an excessive flange friction on the inner rail and had bound the trucks by keeping the inner side bearings in continuous contact. Since this component is opposed by the centrifugal force developed by the car in rounding the curve, these effects should continuously diminish as the centrifugal force increases up to the speed where this component of car weight equals the centrifugal force. Here the point of minimum resistance would

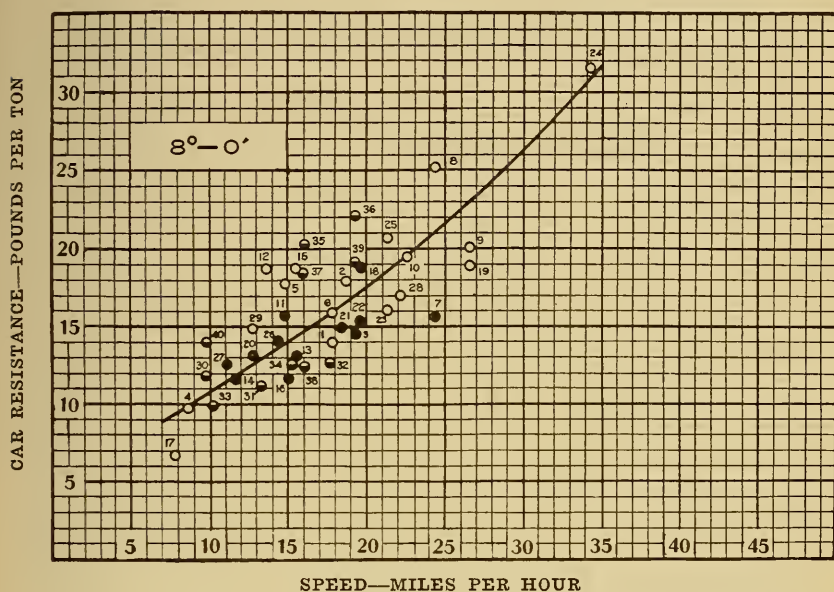


FIG. 12. THE RELATION OF RESISTANCE TO SPEED ON THE 8°-0' CURVE

be expected to occur. Calculations for this case, however, show this speed to be at 16 instead of at $12\frac{1}{2}$ miles per hour. This discrepancy is not in itself enough to discredit the assumption; but investigation of the other curves shows that this speed, at which the centrifugal force and the weight component are in balance, occurs at from 23 to 26 miles per hour, and, if the assumption were valid, similar minimum points should occur at these speeds in the resistance curves for these cases. No such minimum points occur, however, in any of the other figures, and the explanation on these grounds must consequently be abandoned. The condition presented in Fig. 13 is accepted as unexplained by the information at hand.

From the results shown in Figs. 3 to 13, grade and acceleration resistance have been eliminated, and, as previously explained, the effect of the component of wind velocity parallel to the track has also

been eliminated; the influence of the normal component of wind velocity, on the other hand, is still embodied in these results. The resistance defined by the line drawn in Fig. 3, for example, which relates to the 2-degree curve, consequently comprises net resistance on level track at uniform speed, resistance due to the component of wind velocity normal to the track, and the resistance due to curvature. The resistance defined by the line in Fig. 4, which relates to the tan-

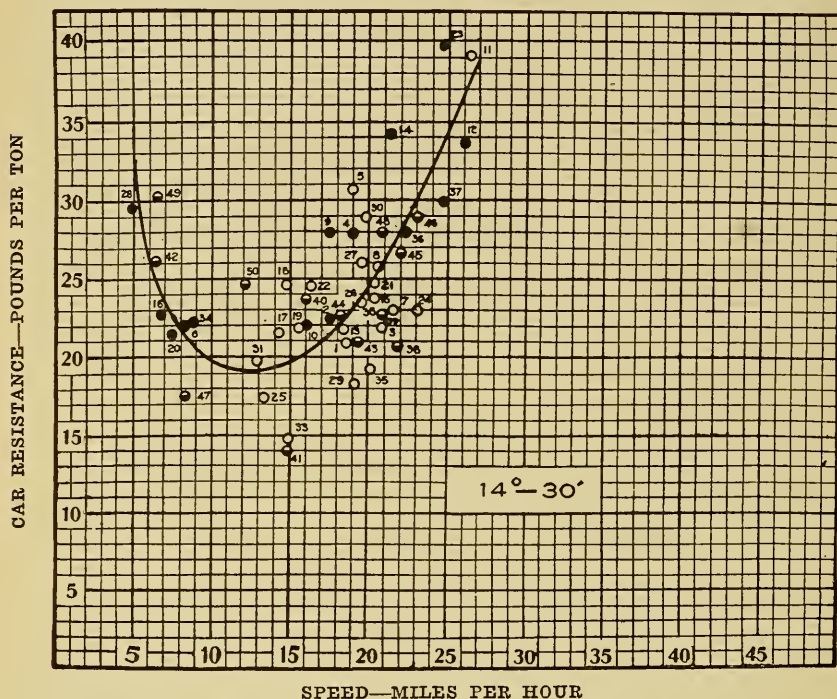


FIG. 13. THE RELATION OF RESISTANCE TO SPEED ON THE 14°-30' CURVE

gent adjoining the 2-degree curve, comprises net resistance on level track at uniform speed and the resistance due to the normal component of wind velocity. The ordinates of the curves in these two corresponding figures, therefore, differ only by an amount equal to the resistance due to curvature on the 2-degree curve. Similar statements apply to the curves in Figs. 5 to 13.

In the further consideration of Figs. 3 to 13 it will be convenient to keep in mind their relationship. The 2°-0', 2°-50', 3°-40', and 6°-30' curves (Figs. 3, 5, 7, 10) have each their own tangent (Figs. 4, 6, 8, 11), the diagram for which immediately follows the diagram for the corresponding curve. The 5°-0', 8°-0', and 14°-30' curves (Figs.

9, 12, 13) have all the same tangent as the 3° -40' curve. The relation between these figures is as follows:

- Fig. 3, the 2° -0' curve, is used with Fig. 4, Tangent W.
- Fig. 5, the 2° -50' curve, is used with Fig. 6, Tangent S.
- Fig. 7, the 3° -40' curve, is used with Fig. 8, Tangent R.
- Fig. 9, the 5° -0' curve, is used with Fig. 8, Tangent R.
- Fig. 10, the 6° -30' curve, is used with Fig. 11, Tangent D.
- Fig. 12, the 8° -0' curve, is used with Fig. 8, Tangent R.
- Fig. 13, the 14° -30' curve, is used with Fig. 8, Tangent R.

VI. THE FINAL RESULTS OF THE TESTS

9. *The Resistance Due to Curvature.*—The resistance-speed curves of Figs. 3 to 13, properly paired, are all brought together in Fig. 14 in which seven pairs of lines appear, one for each of the seven curves. The first pair of lines in Fig. 14 relates to the 2-degree curve, the upper line marked *C* shows the relation between resistance and speed on the curve, while the lower line marked *T* shows this relation on the adjacent tangent. These lines are reproduced from Figs. 3 and 4, respectively. Fig. 14 presents also six additional pairs of curves similarly marked pertaining to the six remaining curves, and all reproduced from Figs. 5 to 13.

As has been stated in Chapter V the difference between the ordinates of these pairs of lines at any speed is the value of curve resistance at that speed for the curve in question. For example, for the 2-degree curve (see Fig. 14) the resistance on the curve at 20 miles per hour is 14.0 pounds per ton, and the resistance on the tangent is 11.3 pounds per ton. The difference between these values, 2.7 pounds per ton, is the curve resistance at 20 miles per hour on the 2-degree curve. At speeds varying from 10 to 40 miles per hour the values of resistance on the curve, resistance on the tangent, and curve resistance have been thus determined for each of the seven curves and set forth in Table 3. This table summarizes, therefore, the direct results of the tests as presented in Figs. 3 to 14, inclusive, and it presents also the derived values of curve resistance whose determination was the immediate purpose of the tests. The values of curve resistance given in the table are accepted as the average values at the various speeds. As has been previously explained these values are nearly, if not quite, freed from any effect of wind.

10. *The Relation of Curve Resistance to Curvature.*—For each of the speeds in Table 3 there appears for each of the seven curves a

TABLE 3

VALUES OF RESISTANCE ON THE CURVE, RESISTANCE ON THE TANGENT, AND RESISTANCE DUE TO CURVATURE FOR EACH OF THE SEVEN CURVES AND AT VARIOUS SPEEDS. THESE VALUES ARE DERIVED DIRECTLY FROM FIGURES 3 TO 13 INCLUSIVE. THEY ARE EXPRESSED IN POUNDS PER TON

Curve	Speed—M.P.H.							
	10	15	20	25	30	35	40	
2°-0'	7.50	10.78	14.00	17.45	21.00	24.70	28.70	Res. on Curve
	6.95	9.00	11.30	13.93	16.88	20.25	24.15	" " Tang.
	0.55	1.78	2.70	3.52	4.12	4.45	4.55	Curve Res.
2°-50'	12.20	13.40	15.05	17.30	20.00	23.25	27.00	Res. on Curve
	6.95	8.55	10.44	12.75	15.30	18.21	21.50	" " Tang.
	5.25	4.85	4.61	4.55	4.70	5.04	5.50	Curve Res.
3°-40'	8.00	10.50	13.25	16.40	20.50	26.20	33.00	Res. on Curve
	5.80	8.02	10.60	13.45	16.70	20.47	24.50	" " Tang.
	2.20	2.48	2.65	2.95	3.80	5.73	8.50	Curve Res.
5°-0'	12.25	15.00	18.33	22.30	26.83	31.62	36.70	Res. on Curve
	5.80	8.02	10.60	13.45	16.70	20.47	24.50	" " Tang.
	6.45	6.98	7.78	8.85	10.13	11.15	12.20	Curve Res.
6°-30'	10.60	14.65	18.75	23.00	27.50	32.05	36.55	Res. on Curve
	6.87	8.62	10.55	12.65	15.00	17.80	20.95	" " Tang.
	3.73	6.03	8.20	10.35	12.50	14.25	15.60	Curve Res.
8°-0'	10.80	14.05	17.55	21.60	26.25	31.75		Res. on Curve
	5.80	8.02	10.60	13.45	16.70	20.47		" " Tang.
	5.00	6.03	6.95	8.15	9.55	11.28		Curve Res.
14°-30'		19.70	24.60	34.45				Res. on Curve
		8.02	10.60	13.45				" " Tang.
		11.68	14.00	21.00				Curve Res.

value of curve resistance. At 15 miles per hour, for example, there are given in the third column of the table seven values of curve resistance: 1.78 pounds per ton on the 2-degree curve, 4.85 pounds per ton on the 2°-50' curve, and so on. These values are plotted in the second diagram of Fig. 15 as seven points which show the relations existing during the tests between curve resistance and curvature at the speed of 15 miles per hour. Using the remaining values from Table 3, similar groups of points have been plotted in Fig. 15 which show this relation at the other speeds.

An inspection of the points in Fig. 15 discloses at all speeds a fairly regular relationship between curve resistance and curvature. With perhaps the exception of the first, pertaining to 10 miles per hour, these diagrams show for each speed an increase in curve resistance which is approximately directly proportional to the increase in curvature. There have been drawn among the points in these diagrams straight lines which have been accepted as defining the average of the relations between curve resistance and curvature represented by the points derived from the individual tests. For obvious reasons these lines are all made to pass through the origin of coördinates.

At speeds up to 25 miles per hour the points in Fig. 15 which relate to the $2^{\circ}-50'$ and to the 5-degree curves lie above the mean values represented by the lines, while at higher speeds they correspond closely with the mean. At speeds up to 25 miles per hour the points for the $3^{\circ}-40'$ and for the 8-degree curves correspond well with the mean, but

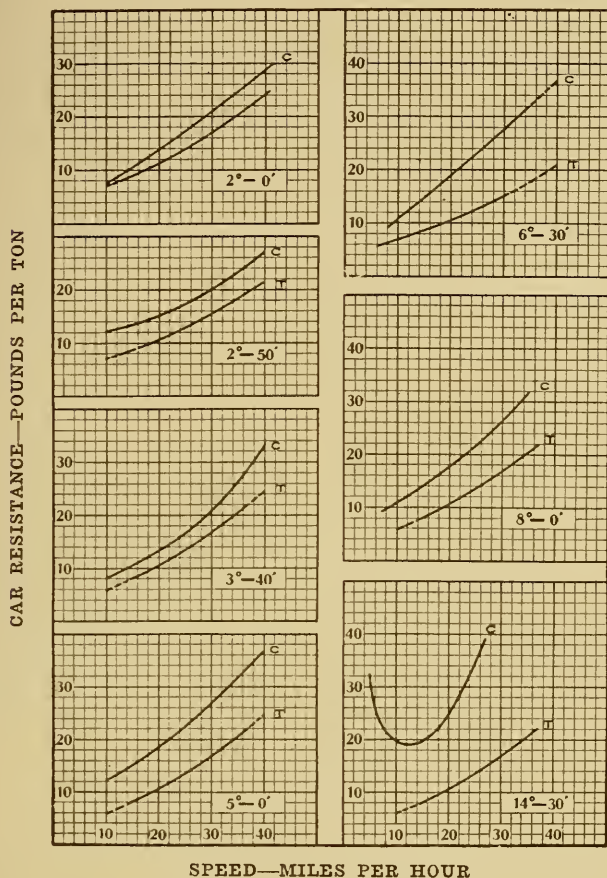


FIG. 14. THE RELATION OF RESISTANCE TO SPEED ON ALL THE CURVES AND ON THEIR CORRESPONDING TANGENTS

lie below it at higher speeds. The test data offer a probable explanation of these variations only in the case of the 5-degree curve, whose poorer construction might be assumed partially to account for its high resistance. In the other cases, however, no such explanation is available. In drawing the lines in Fig. 15 due consideration was given to the relative weight of the points there plotted; that is, to the number of tests represented by the various points.

The lines drawn in Fig. 15 are brought together and reproduced to a larger scale in Fig. 16, in which, as before, each line defines for a particular speed the average or general relation between curve resistance and curvature. The whole group defines this relation for the

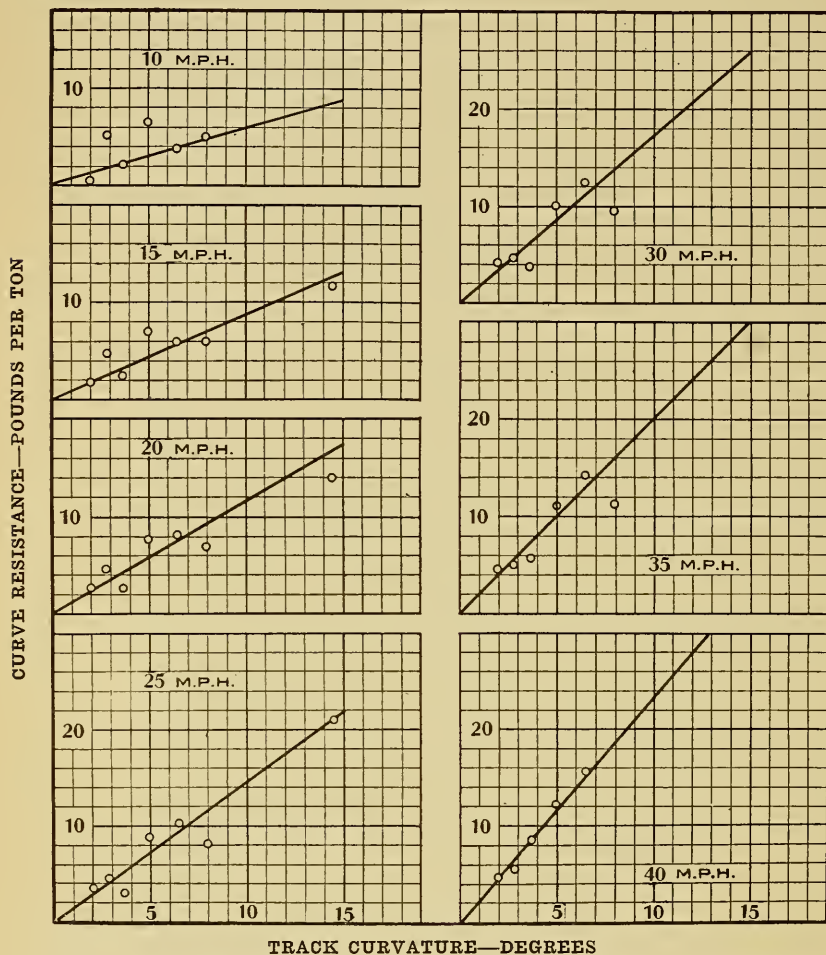


FIG. 15. THE RELATION BETWEEN CURVE RESISTANCE AND CURVATURE AT VARIOUS SPEEDS

entire series of tests. For each speed curve resistance increases in direct proportion with the curvature. At 15 miles per hour, for example, the curve resistance expressed in pounds per ton is 4.35, 8.70, and 13.05, for curves of 5, 10, and 15 degrees, respectively. This implies at this speed a constant curve resistance of 0.87 pounds, when expressed in pounds per ton *per degree*. The direct proportionality

between curve resistance and curvature exhibited in Fig. 16 is in accord with the results of previous experiments and with current practice.

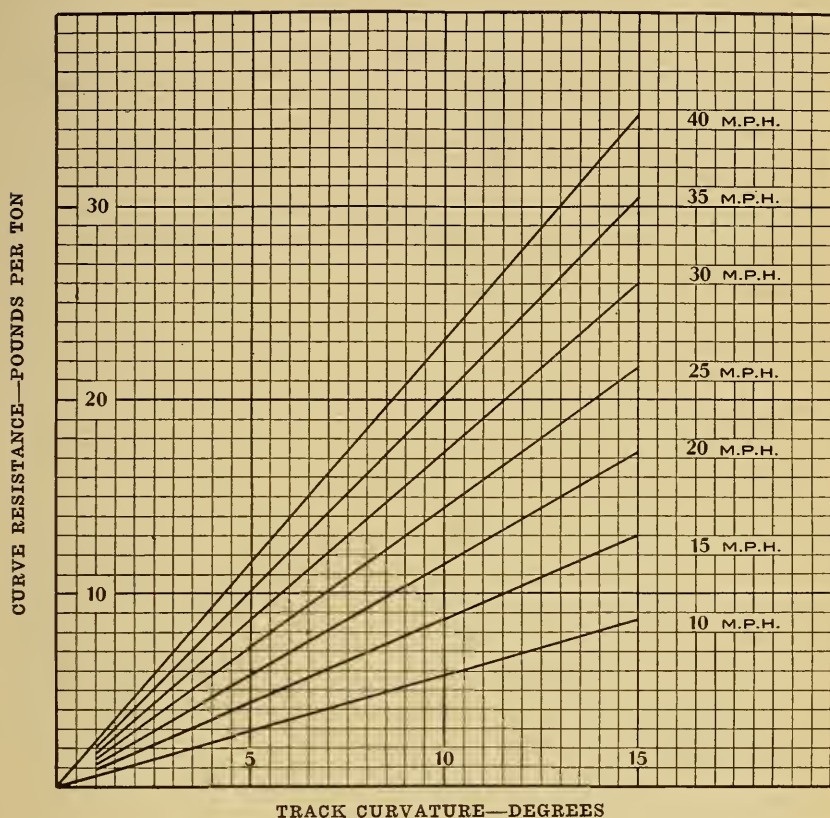


FIG. 16.—THE RELATION BETWEEN CURVE RESISTANCE AND CURVATURE, AT VARIOUS SPEEDS. THE LINES SHOWN ARE ASSEMBLED FROM FIG. 15

11. *The Relation of Curve Resistance to Speed.*—The lines of Fig. 16 present values of curve resistance for seven speeds which cover the speed range of the experiments. The figure offers, therefore, a means of determining the general relation between curve resistance and speed. If in Fig. 16 the ordinates of the seven lines are measured at a curvature of 5 degrees, seven values of curve resistance are obtained: 2.90, 4.35, 5.80, 7.25, 8.70, 10.15, and 11.60 pounds per ton, which correspond respectively to speeds of 10, 15, 20, 25, 30, 35, and 40 miles per hour. These corresponding values of resistance and speed are plotted in Fig. 17 as the seven points there shown for a curvature of 5 degrees. These points are found to lie on a straight line as shown

in the figure. The two other straight lines corresponding to 10 degrees and 15 degrees curvature were obtained by a like process. That the lines connecting the points in Fig. 17 are straight is due, of course, to the relation which exists between the lines in Figs. 15 and 16; that is, to their relative slope. It is proper to explain that the lines originally drawn in Figs. 15 and 16 did not precisely satisfy this condition in the derived lines of Fig. 17. They did so so nearly, however, that, for the sake of the resulting simplicity, it seemed justifiable to modify them, and the slopes of a few of the lines in Fig. 15

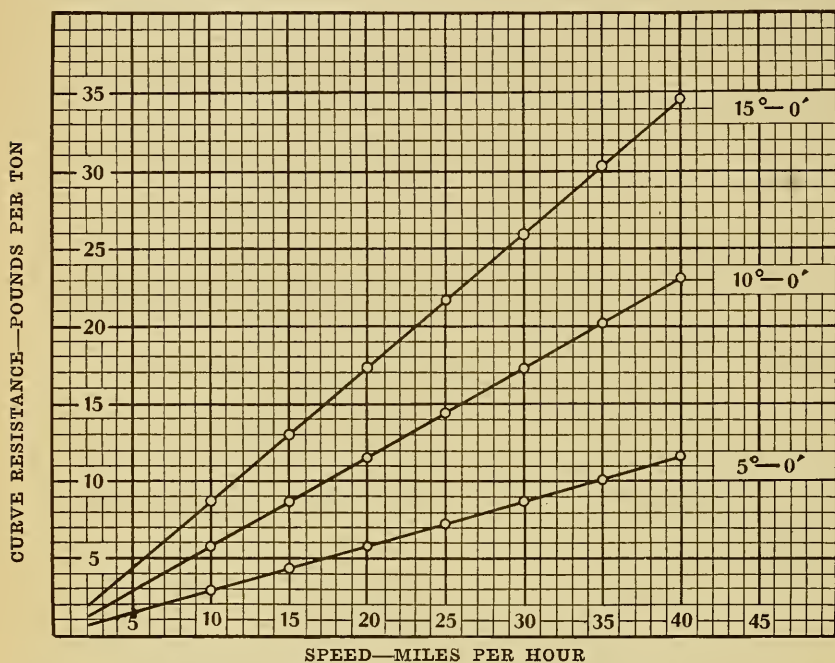


FIG. 17.—THE RELATION BETWEEN CURVE RESISTANCE AND SPEED FOR CURVES OF VARIOUS CURVATURES

were adjusted so that the lines connecting the derived points in Fig. 17 would be straight. An inspection of Fig. 15 shows, it is believed, that this adjustment has done no violence to the experimental data. It has, on the other hand, resulted in a simplicity of the relations shown in Fig. 17 and of the formula given beyond, which seems amply to warrant it.

The lines drawn in Fig. 17 apply to curvatures of 5, 10, and 15 degrees, and by the process above explained similar lines for other rates of curvature might have been included. Fig. 17 is derived directly from Fig. 16 and, like it, presents the average results of the

whole research. From Fig. 17 it is obvious that for constant curvature, curve resistance was directly proportional to speed; that is, on any given curve the resistance due to curvature varied in these experiments directly with the speed. On the 5-degree curve, for example, its average value at 10 miles per hour was 2.90 pounds per ton, and at 30 miles per hour 8.70 pounds per ton, three times as much. In previous experiments there has been occasional evidence that curve resistance might be greater at high than at low speeds, although in some discussions of the subject the contrary is held to be true. Neither opinion has had definite or adequate support, and there has hitherto been no conclusive experimental evidence of any definite relation be-

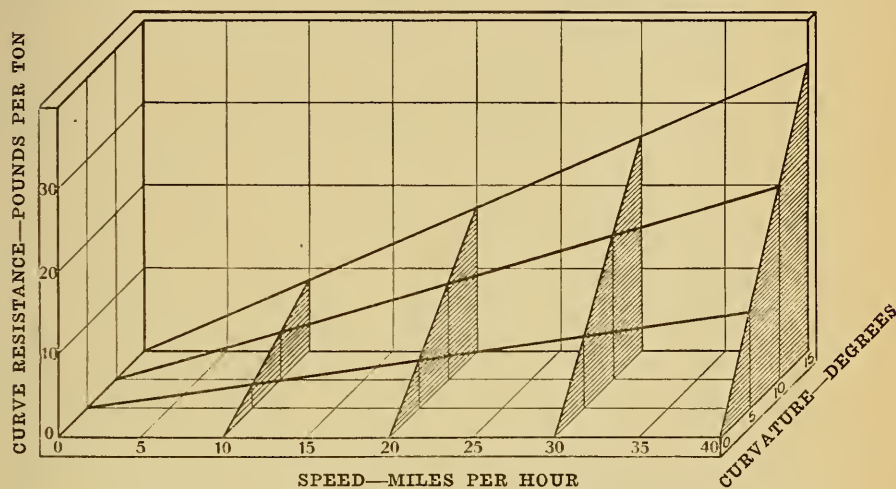


FIG. 18. THE CONCURRENT RELATIONS BETWEEN CURVE RESISTANCE, SPEED, AND CURVATURE

tween curve resistance and speed. In practice the influence of speed has been generally ignored.

12. *The Concurrent Relations of Curve Resistance, Curvature, and Speed.*—For the car used in the tests curve resistance varies directly with both curvature and speed, the rate of variation in each case being as shown in Figs. 16 and 17. These two figures have been combined in the diagram of Fig. 18, which exhibits the concurrent relations of the three variables. These relations are also defined by the equation:

$$R_c = 0.058 S C \dots \dots \dots (1)$$

in which R_c is the curve resistance on level track at uniform speed expressed in pounds per ton, S is the speed in miles per hour, and C is the degree of curve. This formula represents exactly the mean

relations between curve resistance, speed, and curvature, which are defined by the lines drawn in Figs. 15, 16, and 17, and it embodies, consequently, the generalized results of all the tests.

By means of Formula 1 there have been calculated the values of curve resistance which are presented in Table 4. The values are given for curves varying from 1 to 15 degrees and for speeds ranging from 10 to 40 miles per hour, corresponding approximately to the range in curvature and in speed during the tests. Inspection of Table 4 shows curve resistance to vary from 0.58 pounds per ton on a 1-

TABLE 4

VALUES OF CURVE RESISTANCE AT VARIOUS RATES OF CURVATURE AND AT VARIOUS SPEEDS. THESE VALUES ARE DERIVED FROM FORMULA 1 AND REPRESENT THE FINAL RESULTS OF THE TESTS

Curvature Degrees	Curve Resistance — Pounds Per Ton							Curvature Degrees
	Column Headings Indicate Speed in Miles Per Hour							
	10	15	20	25	30	35	40	
1	0.58	0.87	1.16	1.45	1.74	2.03	2.32	1
2	1.16	1.74	2.32	2.90	3.48	4.06	4.64	2
3	1.74	2.61	3.48	4.35	5.22	6.09	6.96	3
4	2.32	3.48	4.64	5.80	6.96	8.12	9.28	4
5	2.90	4.35	5.80	7.25	8.70	10.15	11.60	5
6	3.48	5.22	6.96	8.70	10.44	12.18	13.92	6
7	4.06	6.09	8.12	10.15	12.18	14.21	16.24	7
8	4.64	6.96	9.28	11.60	13.92	16.24	18.56	8
9	5.22	7.83	10.44	13.05	15.66	18.27	20.88	9
10	5.80	8.70	11.60	14.50	17.40	20.30	23.20	10
11	6.38	9.57	12.76	15.95	19.14	22.33	25.52	11
12	6.96	10.44	13.92	17.40	20.88	24.36	27.84	12
13	7.54	11.31	15.08	18.85	22.62	26.39	30.16	13
14	8.12	12.18	16.24	20.30	24.36	28.42	32.48	14
15	8.70	13.05	17.40	21.75	26.10	30.45	34.80	15

degree curve at 10 miles per hour to 34.8 pounds per ton on a 15-degree curve at 40 miles per hour. Expressed in pounds per ton *per degree of curve*, curve resistance varies from 0.58 at 10 miles per hour to 2.32 at 40 miles per hour. Although for the sake of exact correspondence between the table and Formula 1 and Figs. 16 and 17 the tabular values are given to the second decimal place, it should not be assumed that the last figures are significant.

13. *Conclusion.*—The final results of the tests are summarized in Formula 1 and in Table 4, which present mean or average values of curve resistance at various rates of curvature and at various speeds. They apply to the particular car tested under conditions of weather and track which have been fully defined. They are useful for predicting the curve resistance to be expected from similar cars running under like conditions, but if they are so used these conditions should be borne in mind. It is probable that the formula applies to cars

whose weight varies considerably from the weight of the test car; but if the truck wheel base is materially different, the formula should be used with caution. Its use should probably not be extended to speeds much in excess of 40 miles per hour. How far the formula is applicable beyond a curvature of 15 degrees cannot be stated, but it is significant that it represents the actual test results on the 14½-degree curve with considerably greater accuracy than on some of the curves of lower curvature. Whether the results apply closely to other than self-propelled cars is open to question.

Formula 1 and Table 4, as well as Figs. 16 and 17, are generalizations which define only average values of curve resistance. They are derived by processes which preclude their precise agreement even with all the test values themselves, as is disclosed by an examination of Fig. 15. Consequently, in an individual case their use in predicting curve resistance cannot reasonably be expected to give results which correspond with the actual resistance any more exactly than do the averages upon which these generalizations rest correspond with the results of the individual tests; no closer correspondence, that is, should be expected than exists between the mean curves and the points of Figs. 3 to 13 which are fundamental to the whole process. Such limitations are common to all generalizations of train or car resistance and do not seriously impair their usefulness for the purposes for which they are generally employed.

Bulletin 74 of the Engineering Experiment Station gives the results of experiments in which the resistance of this car was determined when running on level tangent track at uniform speed. This resistance is given by the formula:

$$R_t = 4 + 0.222S + 0.00181 \frac{A}{W} S^2 \dots\dots\dots (2)$$

By combining this equation with Formula 1 above, we obtain for this car the following formula which gives its total resistance when running on a level curve at uniform speed.

$$R = R_t + R_c = 4 + 0.222S + 0.00181 \frac{A}{W} S^2 + 0.058 SC \dots\dots\dots (3)$$

In Formula 3, R is the total resistance expressed in pounds per ton, S the speed in miles per hour, C the curvature in degrees, A the cross-sectional area of the car in square feet, and W its weight in tons. If the car is on a grade and its speed changing, the usual corrections for grade and acceleration must be applied to Formula 3.

APPENDIX I

THE TRACK

Information regarding the track has been given in Chapter III. This appendix is intended to supplement Chapter III by the presentation of further details concerning the track on the curves.

The results of the track surveys are presented in Figs. 19 to 25, inclusive, in which are given for each curve the profile on each rail, elevations having been taken every 100 feet; the superelevation of the outer rail, also at 100-foot intervals; and the degree of curvature. For the $2^{\circ}-0'$, $2^{\circ}-50'$, $3^{\circ}-40'$, and $5^{\circ}-0'$ curves the curvature was determined at 50-foot intervals by measuring mid-ordinates, for the $6^{\circ}-30'$ curve it was determined at 50-foot intervals by measuring deflection angles, and for the $8^{\circ}-0'$ and $14^{\circ}-30'$ curves by measuring deflection angles at intervals of 25 feet. Each curve has been designated by the average of these values of curvature. Distances are shown in the figures by means of stations which were 100 feet apart. In the tables given in Appendix III the section limits are defined by trolley line pole numbers. The location of each pole is indicated on the alignment diagrams of Figs. 19 to 25 by means of a short dash beside which is recorded its distance in feet from the preceding station. For example, Pole 20, Fig. 19, is located 80 feet from Station 23.

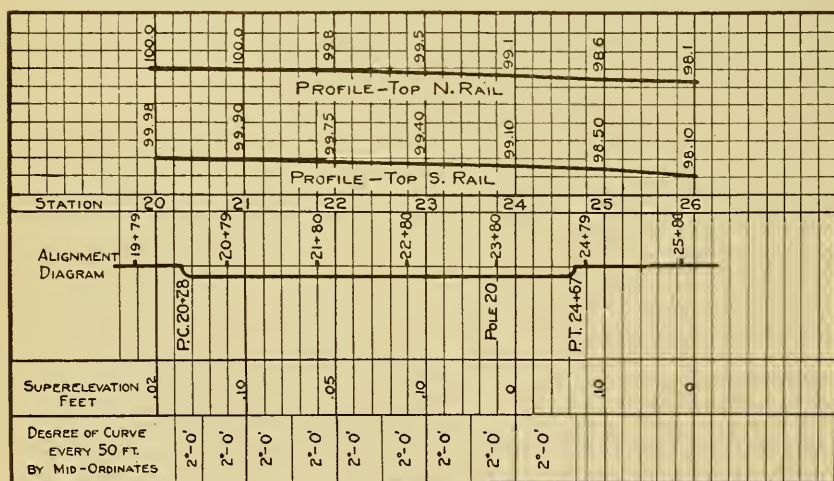


FIG. 19. PROFILE AND ALIGNMENT DIAGRAM OF THE TRACK ON THE $2^{\circ}-0'$ CURVE

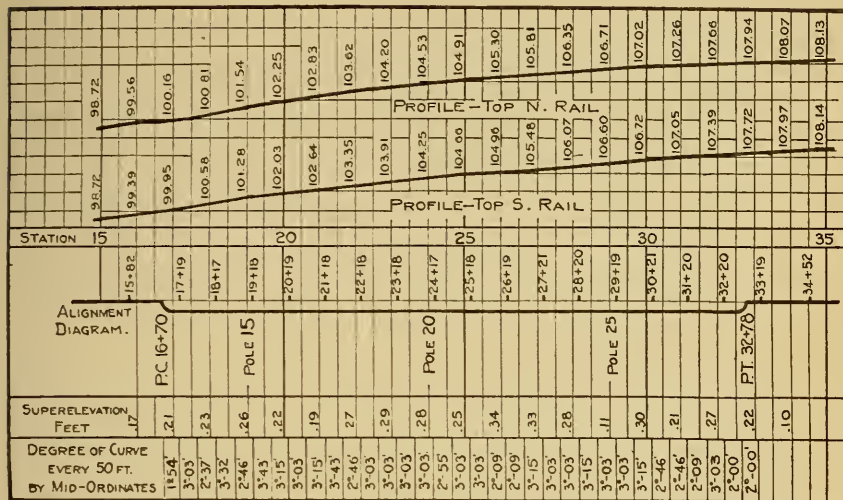


FIG. 20. PROFILE AND ALIGNMENT DIAGRAM OF THE TRACK ON THE 2°-50' CURVE

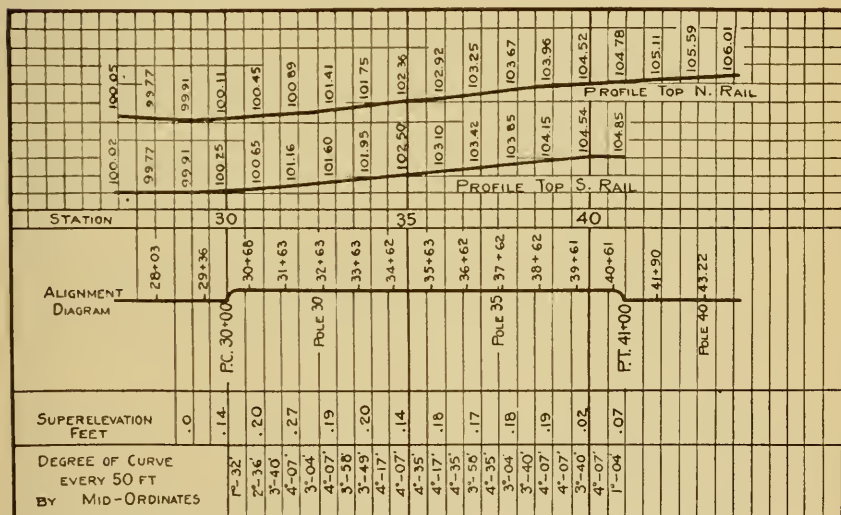


FIG. 21. PROFILE AND ALIGNMENT DIAGRAM OF THE TRACK ON THE 3°-40' CURVE

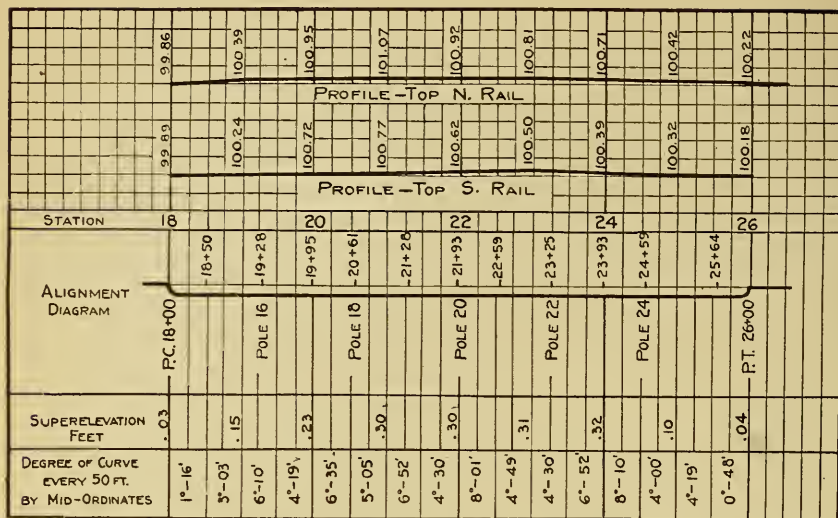


FIG. 22. PROFILE AND ALIGNMENT DIAGRAM OF THE TRACK ON THE 5°-0' CURVE

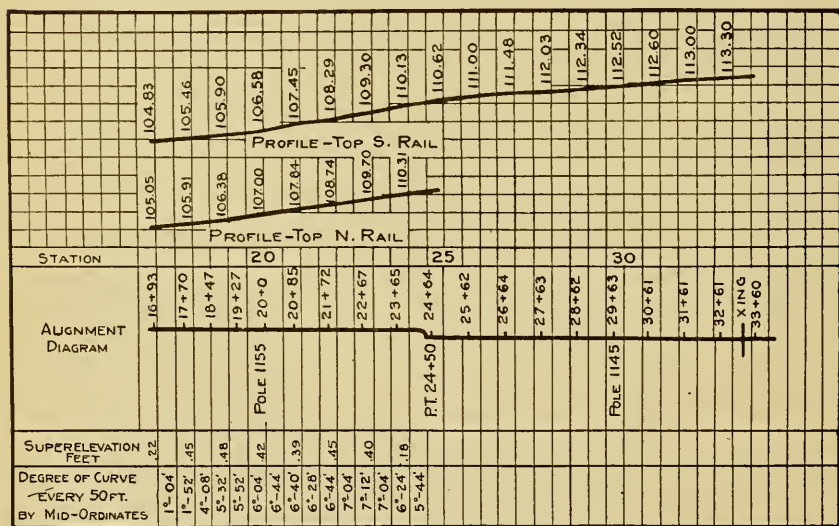


FIG. 23. PROFILE AND ALIGNMENT DIAGRAM OF THE TRACK ON THE 6°-30' CURVE

APPENDIX II

TEST METHODS AND METHODS OF CALCULATION

This appendix is intended to supplement what has been stated in Chapter IV concerning the test methods and to present also certain details concerning the methods used in calculating the values of resistance and speed which constitute the immediate results of the tests.

Methods Used in Producing the Curves of Figs. 3 to 13.—As has been previously explained, the points in Figs. 3 to 13 occur in pairs, one point in each pair relating to a run in which the wind helped the car, the other to a run in which the wind opposed the car. The curves drawn in the figures define the mean of the values represented by these points. If wind resistance varied directly with speed, an arithmetical mean or average of the values represented by the points would define a point on this curve. Since wind resistance varies, however, about as the square of the speed, a mere arithmetical mean of these values does not serve exactly to define the mean curve; but this mean must be established by another and somewhat more elaborate process. Such a process has been set forth on page 34 of Bulletin 74 of the University of Illinois Engineering Experiment Station. When the wind velocities are low, the difference between the curves defined by taking an arithmetical mean and by this more exact process is negligible. During these tests the wind velocities were generally low, and the points representing runs with and against the wind are generally closely interwoven. For this reason the mean curves in Figs. 3 to 13 were all produced by taking the arithmetical mean of the resistance values represented by the points in the figures. Preliminary to drawing these curves, the average coördinates of various groups of points in each figure were determined, and these average coördinates were then plotted as auxiliary points through which the curve was passed as nearly as possible.

Methods of Calculation.—The data recorded during these tests are enumerated in Chapter III. All the data which pertained directly to car operation were graphically recorded upon the test car charts. A portion of the chart obtained during Test 121 is reproduced as Fig. 26, upon which the section limits and certain explanatory lettering have been added.

When the track sections were surveyed, markers were placed beside the track at the ends of the curves and tangents, and during a test the positions of these numbered markers were recorded upon the chart. In this way the chart made during a run over the test track was divided into two or more sections as illustrated by Fig. 26. That

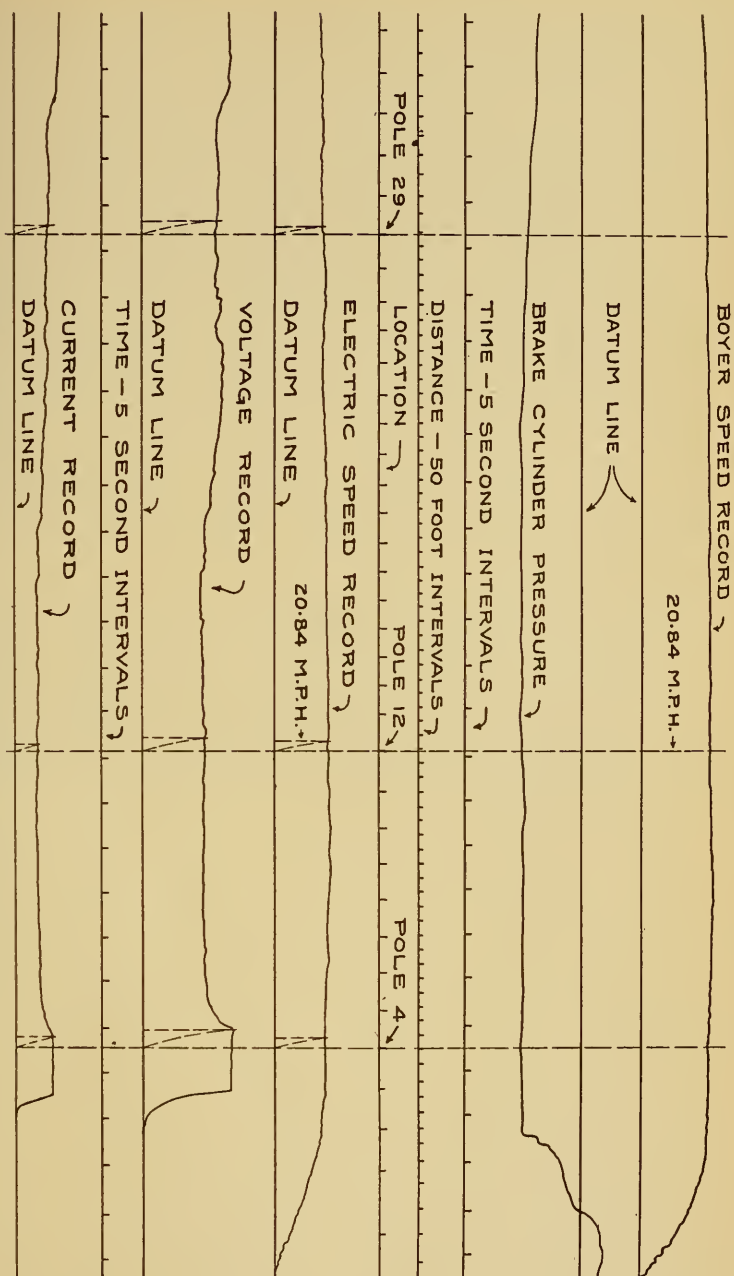


FIG. 26. REPRODUCED FROM A PORTION OF ONE OF THE TEST CAR CHARTS

part of the chart lying between the poles numbered 12 and 29 was produced during a run over a section of curved track, while that portion lying between the poles numbered 4 and 12 was made during a run over a section of tangent track. In making the calculations only those chart sections were considered over which there were no brake applications, no large variation of the current or voltage from the average value, and no large difference in the speed of the car. It has been found desirable to choose both track and chart sections so that the energy consumed by grade and acceleration will be as small as possible, and the errors in their calculation will have slight effect on the accuracy of the values of average net car resistance. When the variation in speed is large the energy required to produce acceleration is alone frequently greater than that required to overcome all other resistances combined. In all the tests reported in this bulletin the variations in speed in passing the track sections have exceeded 2 miles per hour in only 11 per cent of the total number of resistance determinations, and in only 3 cases out of 392 has this speed variation exceeded 5 miles per hour. The maximum variation over any section was less than 10 miles per hour.

The result desired from each chart section is a value of average net car resistance. These values are given in Column 18 of Tables 5 to 11 in Appendix III. Full explanations of the significance of the various items in these tables and of the methods by which they were derived are presented in Appendix II of Bulletin 74 of the Engineering Experiment Station.

APPENDIX III

THE IMMEDIATE RESULTS OF THE TESTS

This appendix presents in tabular form the fundamental data, some of the intermediate results, and the final values of car resistance and speed for each of the curves. The corresponding data for tangents D, W, S, and R, are given in Tables 9, 10, 11, and 12, respectively, of Bulletin 74 of the Engineering Experiment Station. The items in these tables have all been derived by the processes explained in Appendix II. The points in Fig. 3 to 13 were plotted by using as coördinates the values of resistance and speed which appear in Columns 18 and 19 of these tables.

TABLE 5

THE IMMEDIATE RESULTS OF THE TESTS ON THE 2°-0' CURVE, GIVING THE RESULTS OF RESISTANCE AND SPEED USED IN PRODUCING FIG. 3

1	2	3	4	5	6	7	8	9
Item No.	Test No.	Wind Opposing or Helping	Section Limits — Trolley Line Pole Numbers	Grade	Length of Track Section	Time to Run Over Section	Motor Data	
				Rise or Fall Over Section — +Up —Down			Number in Use and Connection *	Efficiency of Motors and Gears
		O or H		Feet	Feet	Sec.		Per cent
1	124	H	16-21	—1.47	500	9.8	4M	73.1
2	123	O	21-16	+ "	"	14.8	4S	73.0
3	123	O	21-16	+ "	"	27.6	"	70.0
4	124	H	16-21	— "	"	10.5	4M	72.2
5	123	O	21-16	+ "	"	9.5	"	81.4
6	124	H	16-21	— "	"	11.3	"	77.6
7	124	H	16-21	— "	"	10.6	"	69.8
8	123	O	21-16	+ "	"	14.2	4S	71.5
9	124	H	16-21	— "	"	14.6	"	67.1
10	117	H	21-16	+ "	"	12.7	"	77.9
11	124	H	16-21	— "	"	27.6	"	64.7
12	123	O	21-16	+ "	"	28.6	"	66.5
13	124	H	16-21	— "	"	16.1	"	72.7
14	123	O	21-16	+ "	"	23.3	"	68.4
15	124	H	16-21	— "	"	27.6	"	63.8
16	118	O	16-21	— "	"	10.5	4M	83.2
17	118	O	16-21	— "	"	19.1	4S	76.7
18	117	H	21-16	+ "	"	9.9	4M	85.2
19	118	O	16-21	— "	"	9.9	"	85.7
20	117	H	21-16	+ "	"	8.7	"	85.0
21	118	O	16-21	— "	"	9.2	"	85.6
22	117	H	21-16	+ "	"	8.2	"	84.3
23	118	O	16-21	— "	"	8.7	"	84.4
24	117	H	21-16	+ "	"	8.6	"	84.5

*S = Series-Multiple. M = Multiple.

TABLE 5 (Continued)

THE IMMEDIATE RESULTS OF THE TESTS ON THE 2°-0' CURVE, GIVING THE RESULTS OF RESISTANCE AND SPEED USED IN PRODUCING FIG. 3

10	11	12	13	14	15	16	17	18	19
Item No.	Speed		Average Voltage	Average Current	Energy Imparted to the Car			Net Car Resistance	Average Speed Over the Section
	At Entrance to the Section	At Exit from the Section			By the Current	By the Change in Kinetic Energy	By the Grade		
	M.P.H.	M.P.H.	Volts	Amp.	Ft. Lb.	Ft. Lb.	Ft. Lb.	Lb. per Ton	M.P.H.
1	33.84	33.84	358	103.0	194820	0	+83420	19.61	34.79
2	23.40	22.50	270	57.6	247840	+84690	—	17.56	23.03
3	10.80	11.34	142	55.7	225410	—24510	—	8.28	12.35
4	32.94	32.76	354	100.0	197920	+24240	+	21.54	32.47
5	33.66	33.66	445	134.8	342130	0	—	18.23	35.88
6	29.70	30.96	378	119.6	292370	—156690	+	15.44	30.17
7	32.94	32.40	342	94.3	175980	+72330	+	23.38	32.16
8	23.58	22.86	268	53.9	216320	+68550	—	14.20	24.01
9	23.04	23.04	229	46.8	154830	0	+	16.79	23.35
10	27.00	26.10	592	53.0	228960	+97010	—82610	17.32	26.84
11	11.16	12.78	121	46.6	148550	—79510	+83420	10.75	12.35
12	11.34	11.34	126	49.4	174630	0	—	6.43	11.92
13	19.98	20.70	245	57.6	243600	—60040	+	18.82	21.17
14	14.40	14.22	157	51.1	188650	+10560	—	8.16	14.64
15	10.80	12.42	122	45.4	143890	—77110	+	10.59	12.35
16	32.04	33.48	470	148.4	449390	—191530	+82610	24.23	32.47
17	18.18	19.08	405	55.1	241150	—68070	+	18.20	17.85
18	35.82	36.18	541	168.4	566700	—52620	—	30.71	34.43
19	34.02	36.00	552	182.8	631350	—281440	+	30.78	34.43
20	40.14	40.68	592	157.4	508190	—88590	—	23.99	39.18
21	36.36	38.34	573	168.5	560760	—300250	+	24.42	37.05
22	42.48	42.12	586	143.6	429000	+61830	—	29.05	41.57
23	39.60	40.32	562	150.8	458950	—116810	+	30.23	39.18
24	39.96	40.14	565	151.4	458440	—29270	—	24.67	39.64

TABLE 6

THE IMMEDIATE RESULTS OF THE TESTS ON THE 2°-50' CURVE, GIVING THE RESULTS OF RESISTANCE AND SPEED USED IN PRODUCING FIG. 5

1	2	3	4	5	6	7	8	9
Item No.	Test No.	Wind Opposing or Helping	Section Limits Trolley Line Pole Numbers	Grade Rise or Fall Over Section +Up —Down	Length of Track Section	Time to Run Over Section	Motor Data	
							Number in Use and Connection *	Efficiency of Motors and Gears
		O or H		Feet	Feet	Sec.		Per cent
1	119	H	29-12	—8.52	1737	39.2	4M	77.4
2	120	O	12-29	+ "	"	38.1	4M	83.7
3	120	O	12-29	+ "	"	41.0	4M	78.4
4	120	O	12-29	+ "	"	46.0	4M	78.9
5	119	H	29-12	— "	"	35.4	4M	67.8
6	120	O	12-29	+ "	"	42.6	4M	82.5
7	119	H	29-12	— "	"	38.3	2M	77.6
8	120	O	12-29	+ "	"	49.4	2M	85.3
9	119	H	29-12	— "	"	49.2	2M	78.2
10	120	O	12-29	+ "	"	54.5	2M	83.1
11	119	H	29-12	— "	"	48.8	4 S	67.5
12	120	O	12-29	+ "	"	61.9	4 S	82.1
13	119	H	29-12	— "	"	50.1	4 S	69.7
14	120	O	12-29	+ "	"	57.7	4 S	82.0
15	119	H	29-12	— "	"	47.7	4 S	72.4
16	120	O	12-29	+ "	"	41.3	4M	80.3
17	120	O	12-29	+ "	"	46.0	2M	85.7
18	120	O	12-29	+ "	"	45.9	2M	85.5
19	119	H	29-12	— "	"	46.0	4 S	71.2
20	120	O	12-29	+ "	"	59.2	4 S	80.7
21	121	O	29-12	— "	"	51.3	4 S	62.8
22	122	H	12-29	+ "	"	57.2	4 S	77.9
23	121	O	29-12	— "	"	52.5	4 S	61.6
24	122	H	12-29	+ "	"	64.3	4 S	76.8
25	121	O	29-12	— "	"	35.3	4M	65.1
26	121	O	29-12	— "	"	36.6	4M	72.0
27	122	H	12-29	+ "	"	61.2	4Sw	76.0
28	121	O	29-12	— "	"	59.0	4Sw	55.6
29	121	O	29-12	— "	"	71.3	2M	73.4
30	121	O	29-12	— "	"	56.0	4 S	64.9
31	122	H	12-29	+ "	"	77.2	4 S	75.1
32	121	O	29-12	— "	"	56.9	4 S	64.1
33	121	O	29-12	— "	"	32.7	4M	77.7
34	122	H	12-29	+ "	"	34.4	4M	83.8
35	121	O	29-12	— "	"	30.1	4M	73.8
36	122	H	12-29	+ "	"	78.2	4 S	75.4
37	121	O	29-12	— "	"	83.9	C	—
38	121	O	29-12	— "	"	106.5	C	—

*S = Series-Multiple. M = Multiple. Sw = Switching. C = Coasting.

TABLE 6 (Continued)

THE IMMEDIATE RESULTS OF THE TESTS ON THE 2°-50' CURVE, GIVING THE RESULTS OF RESISTANCE AND SPEED USED IN PRODUCING FIG. 5

10	11	12	13	14	15	16	17	18	19
Item No.	Speed		Average Voltage	Average Current	Energy Imparted to the Car			Net Car Resistance	Average Speed Over the Section
	At Entrance to the Section	At Exit from the Section			By the Current	By the Change in Kinetic Energy	By the Grade		
	M.P.H.	M.P.H.	Volts	Amp.	Ft. Lb.	Ft. Lb.	Ft. Lb.	Lb. per Ton	M.P.H.
1	26.96	32.18	358	123.2	987020	-632860	+483510	17.00	30.21
2	30.38	31.46	436	159.9	1639720	-136910	—	20.68	31.08
3	31.10	29.12	365	127.8	1105890	+244430	—	17.59	28.89
4	26.42	26.06	340	139.2	1266870	+38730	—	16.68	25.75
5	31.82	33.03	330	90.1	526250	-167640	+	17.09	33.46
6	26.78	27.86	397	152.3	1567090	-120970	—	19.53	27.80
7	31.10	31.10	380	59.5	495590	0	+	19.87	30.92
8	23.90	24.62	417	119.2	1544830	-71620	—	20.08	23.97
9	22.46	25.88	325	69.3	639000	-338910	+	15.90	24.07
10	22.46	21.02	355	101.5	1203440	+128350	—	17.21	21.73
11	23.54	25.52	425	41.7	430480	-199130	+	14.50	24.27
12	19.58	19.76	488	67.8	1240200	-14520	—	15.06	19.13
13	22.10	24.98	438	43.5	490600	-277960	+	14.13	23.64
14	21.92	21.38	516	65.9	1186400	+47930	—	15.24	20.53
15	23.72	26.06	472	46.1	554210	-238790	+	16.22	24.83
16	30.02	29.12	378	138.9	1284060	+109110	—	18.46	28.68
17	24.98	26.24	458	121.1	1612430	-132300	—	20.22	25.75
18	25.16	25.52	443	114.3	1465370	-37400	—	19.16	25.80
19	23.54	26.60	482	44.5	518120	-314530	+	13.94	25.75
20	21.38	19.76	488	62.7	1078150	+136630	—	14.84	20.01
21	22.10	22.10	196	42.4	394880	0	+	17.82	23.09
22	20.84	20.48	278	75.3	1375940	+30490	—	18.72	20.70
23	21.56	21.74	189	41.4	373260	-15980	+	17.06	22.56
24	18.86	18.50	239	74.3	1293620	+27570	—	17.00	18.42
25	33.26	33.08	288	86.5	422170	+24480	+	18.87	33.55
26	31.64	32.54	327	104.3	662910	+118410	+	20.86	32.36
27	18.50	17.78	223	71.8	1098370	+53550	—	13.56	19.35
28	19.76	19.94	160	37.2	287900	+14650	+	15.36	20.07
29	14.90	17.06	203	61.3	480140	-141520	+	16.68	16.61
30	19.58	20.84	200	44.6	478180	-104400	—	17.39	21.15
31	15.08	15.62	200	69.5	1188680	-33980	—	13.62	15.34
32	19.58	20.48	194	43.8	457010	-73910	+	17.58	20.81
33	35.24	36.50	420	113.8	895690	-185300	+	24.22	36.22
34	34.34	32.72	479	155.7	1585580	+222710	—	26.88	34.43
35	38.30	38.66	407	98.9	659400	-56800	+	22.04	39.35
36	12.74	14.18	203	71.1	1255410	-79470	—	14.05	15.14
37	16.34	13.46	..	..	0	+175940	+	13.38	14.12
38	13.10	11.30	..	..	0	+90040	+	11.64	11.12

TABLE 7

THE IMMEDIATE RESULTS OF THE TESTS ON THE 3°-40' CURVE, GIVING THE RESULTS OF RESISTANCE AND SPEED USED IN PRODUCING FIG. 7

1	2	3	4	5	6	7	8	9
Item No.	Test No.	Wind Opposing or Helping	Section Limits Trolley Line Pole Numbers	Grade Rise or Fall Over Section +Up -Down	Length of Track Section	Time to Run Over Section	Motor Data	
							Number in Use and Connection *	Efficiency of Motors and Gears
		O or H		Feet	Feet	Sec.		Per cent
1	126	H	27-38	+4.81	1125	25.3	4M	78.0
2	125	O	38-27	— "	"	24.1	"	66.6
3	125	O	38-27	— "	"	25.2	"	71.0
4	126	H	27-38	+ "	"	28.2	"	79.8
5	126	H	27-38	+ "	"	24.7	"	83.5
6	126	H	27-38	+ "	"	23.8	"	84.2
7	126	H	27-38	+ "	"	38.0	4S	75.5
8	125	O	38-27	— "	"	29.7	"	71.5
9	126	H	27-38	+ "	"	34.6	"	75.7
10	128	..	27-38	+ "	"	36.2	4M	78.1
11	127	..	38-27	— "	"	75.6	"	23.5
12	128	..	27-38	+ "	"	48.7	4S	71.7
13	129	H	38-27	— "	"	29.9	"	48.7
14	130	O	27-38	+ "	"	38.1	"	75.3
15	129	H	38-27	— "	"	40.8	"	35.0
16	130	O	27-38	+ "	"	43.1	"	74.2
17	129	H	38-27	— "	"	20.3	4M	63.3
18	130	O	27-38	+ "	"	23.9	"	82.3
19	129	H	38-27	— "	"	21.1	"	26.7
20	130	O	27-38	+ "	"	24.7	"	78.1
21	130	O	27-38	+ "	"	25.4	"	78.2
22	129	H	38-27	— "	"	25.7	"	30.5
23	130	O	27-38	+ "	"	23.4	"	83.7
24	129	H	38-27	— "	"	26.9	"	64.2
25	130	O	27-38	+ "	"	23.7	"	80.6
26	129	H	38-27	— "	"	55.6	4S	53.3

*S=Series-Multiple. M=Multiple.

TABLE 7 (Continued)

THE IMMEDIATE RESULTS OF THE TESTS ON THE 3°-40' CURVE, GIVING THE RESULTS OF RESISTANCE AND SPEED USED IN PRODUCING FIG. 7

10	11	12	13	14	15	16	17	18	19
Item No.	Speed		Average Voltage	Average Current	Energy Imparted to the Car			Net Car Resistance	Average Speed Over the Section
	At Entrance to the Section	At Exit from the Section			By the Current	By the Change in Kinetic Energy	By the Grade		
	M.P.H.	M.P.H.	Volts	Amp.	Ft. Lb.	Ft. Lb.	Ft. Lb.	Lb. per Ton	M.P.H.
1	32.56	29.68	344	131.1	656380	+371050	—275850	23.30	30.32
2	33.64	32.20	275	90.7	295220	+196260	+	23.79	31.83
3	30.58	30.40	294	104.8	406550	+22720	+	21.86	30.44
4	28.42	26.62	340	147.0	829490	+205080	—	23.52	27.20
5	31.84	32.02	407	162.0	1002830	—23790	—	21.80	31.05
6	32.56	32.56	449	166.9	1107550	0	—	25.78	32.23
7	21.04	20.32	239	67.5	682800	+61640	—	14.52	20.19
8	25.18	26.44	262	54.0	443210	—134640	+	18.12	25.83
9	23.02	20.86	264	66.2	675120	+196200	—	18.46	22.17
10	20.28	20.46	280	152.4	889700	—15220	—276580	18.49	21.19
11	9.66	8.94	91.8	40.4	48610	+27790	+	10.91	10.15
12	17.94	16.68	173	58.4	520450	+90510	—	10.34	15.75
13	26.28	25.02	155	33.1	110180	+133800	+275850	16.12	25.65
14	20.52	20.52	236	66.9	669180	0	—	12.16	20.13
15	18.54	18.18	93.6	26.5	52240	+27360	+	11.02	18.80
16	18.90	18.18	207	64.4	628790	+55260	—	12.65	17.80
17	39.24	37.62	304	81.8	235690	+257740	+	23.85	37.79
18	32.04	30.78	395	151.0	865310	+163850	—	23.35	32.09
19	33.52	36.00	151	43.3	27170	+388730	+	21.44	36.35
20	33.30	31.50	344	132.0	646040	+241440	—	18.96	31.05
21	30.24	28.98	270	156.3	618180	+154460	—	15.40	30.20
22	30.24	28.44	140	47.1	38100	+218640	+	16.51	29.85
23	32.58	32.53	436	160.4	1010100	0	—	22.76	32.78
24	28.80	29.16	222	86.8	245430	—43190	+	14.82	28.51
25	34.74	32.40	383	139.9	754830	+325210	—	24.93	32.86
26	11.16	14.76	103	36.5	164350	—193160	+	7.66	13.80

TABLE 8

THE IMMEDIATE RESULTS OF THE TESTS ON THE 5°-0' CURVE, GIVING THE RESULTS OF RESISTANCE AND SPEED USED IN PRODUCING FIG. 9

1	2	3	4	5	6	7	8	9
Item No.	Test No.	Wind Opposing or Helping	Section Limits Trolley Line Pole Numbers	Grade Rise or Fall Over Section +Up -Down	Length of Track Section	Time to Run Over Section	Motor Data	
							Number in Use and Connection *	Efficiency of Motors and Gears
		O or H		Feet	Feet	Sec.		Per cent
1	125	O	24-14	-0.15	714	14.0	4M	82.9
2	126	H	14-24	+ "	"	15.2	4M	80.5
3	126	H	14-24	+ "	"	17.4	4M	83.1
4	125	O	24-14	- "	"	15.6	4M	77.3
5	126	H	14-24	+ "	"	15.6	4M	85.3
6	126	H	14-24	+ "	"	14.7	4M	83.4
7	126	H	14-24	+ "	"	13.6	4M	85.8
8	126	H	14-24	+ "	"	24.8	4S	76.5
9	125	O	24-14	- "	"	19.0	4S	70.3
10	126	H	14-24	+ "	"	21.4	4S	76.3
11	127	..	24-14	- "	"	25.0	4M	74.9
12	128	..	14-24	+ "	"	23.9	4M	75.8
13	127	..	24-14	- "	"	47.3	4S	70.8
14	128	..	14-24	+ "	"	30.4	4S	73.4
15	129	H	24-14	- "	"	19.9	4S	68.7
16	130	O	14-24	+ "	"	23.7	4S	74.6
17	129	H	24-14	- "	"	27.9	4S	69.9
18	130	O	14-24	+ "	"	31.0	4S	75.0
19	130	O	14-24	+ "	"	15.4	4M	85.5
20	130	O	14-24	+ "	"	14.9	4M	79.0
21	129	H	24-14	- "	"	12.4	4M	83.4
22	130	O	14-24	+ "	"	16.0	4M	85.1
23	130	O	14-24	+ "	"	14.6	4M	82.8

*S=Series-Multiple. M=Multiple.

TABLE 8 (Continued)

THE IMMEDIATE RESULTS OF THE TESTS ON THE 5°-0' CURVE, GIVING THE RESULTS OF RESISTANCE AND SPEED USED IN PRODUCING FIG. 9

10	11	12	13	14	15	16	17	18	19
Item No.	Speed		Average Voltage	Average Current	Energy Imparted to the Car			Net Car Resistance	Average Speed Over the Section
	At Entrance to the Section	At Exit from the Section			By the Current	By the Change in Kinetic Energy	By the Grade		
	M.P.H.	M.P.H.	Volts	Amp.	Ft. Lb.	Ft. Lb.	Ft. Lb.	Lb. per Ton	M.P.H.
1	34.72	35.26	462	146.7	580160	-78220	+8600	24.93	34.77
2	33.10	32.02	383	138.6	479000	+145580	—	30.09	32.03
3	28.42	28.24	387	164.6	679280	+21110	—	33.79	27.98
4	32.74	31.84	354	123.2	387840	+120310	+	25.24	31.21
5	30.22	32.20	460	196.4	886580	-255830	—	30.39	31.21
6	32.74	32.56	436	155.8	614200	+24330	—	30.77	33.12
7	33.82	35.08	524	195.9	883380	-179700	—	33.95	35.80
8	19.60	20.32	243	71.9	488880	-59500	—	20.55	19.63
9	26.26	24.82	245	51.8	250010	+152260	+	20.07	25.62
10	22.30	22.30	268	68.1	439530	0	—	21.05	22.75
11	18.84	19.56	221	131.9	402550	-57370	+8630	17.23	19.47
12	20.10	19.56	245	136.5	446780	+44440	—	23.51	20.37
13	10.74	10.92	121	59.8	357380	-8090	+	17.44	10.29
14	16.32	16.68	189	62.7	390010	-24650	—	17.38	16.01
15	24.30	23.76	220	49.3	218690	+53720	+	13.73	24.46
16	19.98	19.80	230	64.3	385700	+14820	-8600	19.14	20.54
17	17.64	18.54	166	54.0	257380	-67400	+	9.72	17.45
18	16.02	16.74	194	69.9	465020	-48830	—	19.91	15.70
19	30.06	32.58	480	199.6	930380	-326760	—	29.06	31.61
20	33.48	31.86	380	128.4	423550	+219110	—	30.97	32.67
21	39.06	38.88	482	150.6	553640	+29040	+	28.88	39.26
22	27.54	30.24	441	194.1	859580	-322930	—	25.79	30.43
23	34.02	33.66	431	150.0	576390	+50440	—	30.20	33.34

TABLE 9

THE IMMEDIATE RESULTS OF THE TESTS ON THE 6°-30' CURVE, GIVING THE RESULTS OF RESISTANCE AND SPEED USED IN PRODUCING FIG. 10

1	2	3	4	5	6	7	8	9
Item No.	Test No.	Wind Opposing or Helping	Section Limits Trolley Line Pole Numbers	Grade Rise or Fall Over Section +Up -Down	Length of Track Section	Time to Run Over Section	Motor Data	
							Number in Use and Connection *	Efficiency of Motors and Gears
		O or H		Feet	Feet	Sec.		Per cent
1	133	H	1155-1151	+3.22	365	18.3	4S	73.1
2	134	O	1151-1155	- "	"	13.4	"	38.0
3	133	H	1155-1151	+ "	"	25.1	"	74.5
4	134	O	1151-1155	- "	"	17.7	"	49.8
5	133	H	1155-1151	+ "	"	16.5	"	74.5
6	134	O	1151-1155	- "	"	12.7	"	45.1
7	133	H	1155-1151	+ "	"	17.0	"	75.9
8	134	O	1151-1155	- "	"	12.1	"	49.3
9	133	H	1155-1151	+ "	"	20.2	"	75.6
10	133	H	1155-1151	+ "	"	15.0	"	78.3
11	134	O	1151-1155	- "	"	9.1	4M	57.0
12	133	H	1155-1151	+ "	"	9.9	"	83.5
13	134	O	1151-1155	- "	"	7.9	"	54.1
14	134	O	1151-1155	- "	"	7.9	"	76.3
15	134	O	1151-1155	- "	"	13.0	4S	53.7
16	133	H	1155-1151	+ "	"	20.9	"	75.6
17	134	O	1151-1155	- "	"	14.4	"	62.2
18	133	H	1155-1151	+ "	"	12.9	"	78.4
19	134	O	1151-1155	- "	"	11.4	"	50.9
20	133	H	1155-1151	+ "	"	25.7	"	75.2
21	133	H	1155-1151	+ "	"	13.6	"	77.4

*S=Series-Multiple. M=Multiple.

TABLE 9 (Continued)

THE IMMEDIATE RESULTS OF THE TESTS ON THE 6°-30' CURVE, GIVING THE RESULTS OF RESISTANCE AND SPEED USED IN PRODUCING FIG. 10

10	11	12	13	14	15	16	17	18	19
Item No.	Speed		Average Voltage	Average Current	Energy Imparted to the Car			Net Car Resistance	Average Speed Over the Section
	At Entrance to the Section	At Exit from the Section			By the Current	By the Change in Kinetic Energy	By the Grade		
	M.P.H.	M.P.H.	Volts	Amp.	Ft. Lb.	Ft. Lb.	Ft. Lb.	Lb. per Ton	M.P.H.
1	14.59	12.38	173	63.2	215730	+123080	-184510	14.75	13.60
2	18.84	18.84	106	13.2	10510	0	+	18.65	18.57
3	9.49	10.00	146	78.4	315710	-20530	-	10.58	9.91
4	13.23	14.08	97.2	34.6	43730	-47940	+	17.24	14.06
5	15.61	14.25	202	66.3	242830	+83860	-	13.60	15.08
6	18.84	19.52	149	31.4	39530	-53870	+	16.27	19.60
7	14.93	14.25	202	74.4	286030	+40980	-	13.63	14.64
8	19.69	20.03	140	33.7	41510	-27890	+	18.95	20.57
9	12.04	11.53	182	77.5	317720	+24320	-	15.11	12.32
10	16.29	16.46	248	84.5	363020	-11500	-	15.97	16.59
11	27.17	27.17	225	73.8	63500	0	+	23.71	27.35
12	24.45	24.11	380	180.0	417000	+34090	-	25.49	25.14
13	31.76	31.42	230	69.8	50600	+44360	+	26.72	31.50
14	31.59	31.93	365	115.8	187890	-44600	+	31.35	31.50
15	18.84	19.18	158	35.9	58400	-26690	+	20.68	19.14
16	11.19	11.19	184	77.2	331060	0	-	14.01	11.91
17	16.12	17.48	169	42.4	94670	-94360	+	17.68	17.28
18	19.69	19.18	282	77.9	327710	+40940	-	17.61	19.29
19	20.71	21.05	169	34.1	49320	-29320	+	19.56	21.83
20	9.49	11.53	157	83.4	373260	-88550	-	9.58	9.68
21	18.67	17.82	256	75.6	300490	+64050	-	17.21	18.30

TABLE 10

THE IMMEDIATE RESULTS OF THE TESTS ON THE 8°-0' CURVE, GIVING THE RESULTS OF RESISTANCE AND SPEED USED IN PRODUCING FIG. 12

1	2	3	4	5	6	7	8	9
Item No.	Test No.	Wind Opposing or Helping	Section Limits — Trolley Line Pole Numbers	Grade	Length of Track Section	Time to Run Over Section	Motor Data	
				Rise or Fall Over Section			Number in Use and Connection *	Efficiency of Motors and Gears
				+Up —Down				
		O or H		Feet	Feet	Sec.		Per cent
1	142	O	1735 —1737.5	+0.65	176	6.7	4S	80.3
2	142	O	1735 —1737.5	+ "	"	6.4	"	66.8
3	141	H	1737.5—1735	— "	"	6.2	"	41.2
4	142	O	1735 —1737.5	+ "	"	13.9	"	73.3
5	142	O	1735 —1737.5	+ "	"	8.1	"	64.8
6	142	O	1735 —1737.5	+ "	"	6.7	"	78.0
7	141	H	1737.5—1735	— "	"	4.9	4M	21.5
8	142	O	1735 —1737.5	+ "	"	4.9	"	65.4
9	142	O	1735 —1737.5	+ "	"	4.5	"	72.3
10	142	O	1735 —1737.5	+ "	"	5.3	"	73.9
11	141	H	1737.5—1735	— "	"	8.1	4S	33.0
12	142	O	1735 —1737.5	+ "	"	8.8	4M	74.0
13	141	H	1737.5—1735	— "	"	7.7	4S	28.2
14	141	H	1737.5—1735	— "	"	10.3	"	21.2
15	142	O	1735 —1737.5	+ "	"	7.7	4M	75.9
16	141	H	1737.5—1735	— "	"	7.95	4S	49.6
17	142	O	1735 —1737.5	+ "	"	15.3	"	70.1
18	141	H	1737.5—1735	— "	"	6.1	"	52.8
19	142	O	1735 —1737.5	+ "	"	4.5	4M	85.2
20	141	H	1737.5—1735	— "	"	9.4	4S	56.4
21	141	H	1737.5—1735	— "	"	6.5	"	59.1
22	141	H	1737.5—1735	— "	"	6.1	"	62.5
23	142	O	1735 —1737.5	+ "	"	5.6	4M	69.7
24	142	O	1735 —1737.5	+ "	"	3.5	"	86.1
25	142	O	1735 —1737.5	+ "	"	5.6	"	74.6
26	141	H	1737.5—1735	— "	"	8.3	4S	59.7
27	141	H	1737.5—1735	— "	"	10.7	C	
28	142	O	1737.5—1735	+ "	"	5.4	4M	69.6
29	142	O	1735 —1737.5	+ "	"	9.4	"	73.4
30	153	..	1737.5—1735	— "	"	12.2	4S	28.5
31	153	..	1737.5—1735	— "	"	9.0	"	43.8
32	154	..	1735 —1737.5	+ "	"	6.8	"	74.0
33	154	..	1735 —1737.5	+ "	"	11.7	"	74.7
34	153	..	1737.5—1735	— "	"	7.8	"	66.8
35	153	..	1737.5—1735	— "	"	7.4	"	40.4
36	153	..	1737.5—1735	— "	"	6.2	"	51.5
37	154	..	1735 —1737.5	+ "	"	7.5	"	76.1
38	153	..	1737.5—1735	— "	"	7.46	"	37.3
39	153	..	1737.5—1735	— "	"	6.2	"	54.8
40	153	..	1737.5—1735	— "	"	12.2	"	46.7

*S=Series-Multiple. M=Multiple. C=Coasting.

TABLE 10 (Continued)

THE IMMEDIATE RESULTS OF THE TESTS ON THE 8°-0' CURVE, GIVING THE RESULTS OF RESISTANCE AND SPEED USED IN PRODUCING FIG. 12

10	11	12	13	14	15	16	17	18	19
Item No.	Speed		Average Voltage	Average Current	Energy Imparted to the Car			Net Car Resistance	Average Speed Over the Section
	At Entrance to the Section	At Exit from the Section			By the Current	By the Change in Kinetic Energy	By the Grade		
	M.P.H.	M.P.H.	Volts	Amp.	Ft. Lb.	Ft. Lb.	Ft. Lb.	Lb. per Ton	M.P.H.
1	17.52	18.96	290	94.9	218390	-109530	-37570	14.02	17.91
2	18.78	17.88	203	46.7	59780	+ 68790	— "	17.89	18.75
3	19.32	19.32	401	24.1	36420	0	+	14.55	19.35
4	7.62	9.06	128	71.4	137360	- 50080	— "	9.77	8.63
5	16.26	15.18	162	45.5	57060	+ 70800	— "	17.75	14.81
6	17.88	18.42	260	79.3	158950	- 40870	— "	15.83	17.91
7	25.26	24.90	117	43.1	3920	+ 37650	+	15.56	24.49
8	26.34	25.26	236	89.2	49750	+116190	— "	25.24	24.49
9	27.06	26.52	301	109.7	79230	+ 60330	— "	20.05	26.67
10	21.12	20.58	258	120.3	89660	+ 46950	— "	19.47	22.64
11	14.82	14.28	86	27.7	9440	+ 32760	+	15.68	14.81
12	13.92	13.56	171	136.4	112000	+ 20630	— "	18.69	13.64
13	16.08	15.72	59	26.2	4980	+ 23870	+	13.06	15.58
14	11.76	11.40	56	22.6	4060	+ 17380	+	11.60	11.65
15	15.54	15.54	212	145.3	132750	0	— "	18.71	15.58
16	15.90	15.90	110	34.3	21960	0	+	11.70	15.09
17	7.26	8.16	108	58.8	100460	- 28940	— "	6.67	7.84
18	20.40	20.04	166	35.3	27840	+ 30350	+	18.83	19.67
19	25.26	26.34	432	204.6	249930	-116190	— "	18.91	26.67
20	13.02	13.02	97	38.9	29560	0	+	13.20	12.77
21	19.68	19.68	169	39.8	38110	0	+	14.88	18.46
22	20.04	20.04	169	42.7	40570	0	+	15.36	19.67
23	22.74	22.20	236	101.3	68830	+ 50600	— "	16.09	21.43
24	34.08	34.44	539	208.3	249520	- 51430	— "	31.56	34.29
25	21.48	20.94	243	126.7	94860	+ 47760	— "	20.65	21.43
26	14.28	14.28	112	41.2	33730	0	+	14.02	14.46
27	11.94	11.40	0	0	0	+ 26280	+	12.55	11.21
28	23.10	22.56	262	100.0	72620	+ 51410	— "	17.00	22.22
29	12.66	12.66	171	130.0	113120	0	— "	14.85	12.77
30	10.04	9.68	57	26.3	7690	+ 14800	+37640	11.80	9.84
31	13.64	13.64	107	31.6	19650	0	+	11.24	13.33
32	17.42	17.42	220	62.5	102060	0	— "	12.64	17.65
33	10.22	11.66	153	78.1	154050	- 65690	— "	9.96	10.26
34	15.08	15.62	165	48.1	60980	- 34570	+	12.57	15.38
35	16.52	15.80	130	29.9	17130	+ 48520	+	20.27	16.22
36	20.66	20.12	179	34.3	28920	+ 45910	+	22.07	19.35
37	15.62	15.62	207	75.4	131410	0	— "	18.40	16.00
38	16.16	15.98	116	28.9	13750	+ 12060	+	12.45	16.09
39	20.12	19.76	165	36.6	30270	+ 29930	— "	19.20	19.35
40	9.86	9.50	68	33.5	19160	+ 14530	+	14.00	9.84

TABLE 11

THE IMMEDIATE RESULTS OF THE TESTS ON THE 14°-30' CURVE, GIVING THE RESULTS OF RESISTANCE AND SPEED USED IN PRODUCING FIG. 13

1	2	3	4	5	6	7	8	9
Item No.	Test No.	Wind Opposing or Helping	Section Limits Trolley Line Pole Numbers	Grade	Length of Track Section	Time to Run Over Section	Motor Data	
				Rise or Fall Over Section +Up -Down			Number in Use and Connection *	Efficiency of Motors and Gears
		O or H		Feet	Feet	Sec.		Per cent
1	141	O	1735-1732	— 0.17	276	10.1	4 S	74.2
2	142	H	1732-1735	+	"	10.7	"	72.9
3	141	O	1735-1732	—	"	9.0	"	60.7
4	142	H	1732-1735	+	"	9.9	"	78.4
5	141	O	1735-1732	—	"	9.9	"	61.7
6	142	H	1732-1735	+	"	22.7	"	72.4
7	141	O	1735-1732	—	"	8.7	"	74.4
8	141	O	1735-1732	—	"	9.1	"	60.0
9	142	H	1732-1735	+	"	10.7	"	75.0
10	142	H	1732-1735	+	"	11.6	"	77.2
11	141	O	1735-1732	—	"	7.1	4M	84.2
12	142	H	1732-1735	+	"	7.2	"	83.0
13	141	O	1735-1732	—	"	10.2	4 S	72.0
14	142	H	1732-1735	+	"	8.8	4M	80.0
15	141	O	1735-1732	—	"	9.2	4 S	76.8
16	142	H	1732-1735	+	"	27.5	"	72.2
17	141	O	1735-1732	—	"	13.2	"	70.3
18	141	O	1735-1732	—	"	12.7	"	60.6
19	141	O	1735-1732	—	"	12.0	"	69.3
20	142	H	1732-1735	+	"	25.0	"	72.9
21	141	O	1735-1732	—	"	9.2	"	75.7
22	141	O	1735-1732	—	"	11.5	"	76.8
23	142	H	1732-1735	+	"	7.6	4M	78.2
24	141	O	1735-1732	—	"	8.1	4 S	73.8
25	141	O	1735-1732	—	"	14.1	"	69.2
26	141	O	1735-1732	—	"	9.6	"	73.9
27	141	O	1735-1732	—	"	9.6	"	75.0
28	142	H	1732-1735	+	"	38.1	"	71.7
29	141	O	1735-1732	—	"	9.8	"	72.9
30	141	O	1735-1732	—	"	9.5	"	77.7
31	141	O	1735-1732	—	"	14.5	"	66.5
32	142	H	1732-1735	+	"	5.6	4M	86.1
33	141	O	1735-1732	—	"	12.6	4 S	68.7
34	142	H	1732-1735	+	"	21.1	"	75.2
35	141	O	1735-1732	—	"	9.3	"	74.7
36	142	H	1732-1735	+	"	8.4	4M	83.8
37	142	H	1732-1735	+	"	7.6	"	79.9
38	153	..	1735-1732	—	"	8.6	4 S	74.5
39	154	..	1732-1735	+	"	9.0	"	77.4
40	153	..	1735-1732	—	"	11.7	"	63.2
41	153	..	1735-1732	—	"	12.6	"	75.1
42	153	..	1735-1732	—	"	29.0	"	67.7
43	153	..	1735-1732	—	"	9.7	"	71.6
44	153	..	1735-1732	—	"	10.3	"	70.8
45	153	..	1735-1732	—	"	8.5	"	67.7
46	153	..	1735-1732	—	"	8.1	"	77.0
47	153	..	1735-1732	—	"	22.3	"	68.2
48	153	..	1735-1732	—	"	9.0	"	75.9
49	153	..	1735-1732	—	"	28.9	"	48.9
50	153	..	1735-1732	—	"	15.4	"	64.7

*S = Series-Multiple. M = Multiple.

TABLE 11 (Continued)

THE IMMEDIATE RESULTS OF THE TESTS ON THE 14°-30' CURVE, GIVING THE RESULTS OF RESISTANCE AND SPEED USED IN PRODUCING FIG. 13

10	11	12	13	14	15	16	17	18	19
Item No.	Speed		Average Voltage	Average Current	Energy Imparted to the Car			Net Car Resistance	Average Speed Over the Section
	At Entrance to the Section	At Exit from the Section			By the Current	By the Change in Kinetic Energy	By the Grade		
	M.P.H.	M.P.H.	Volts	Amp.	Ft. Lb.	Ft. Lb.	Ft. Lb.	Lb. per Ton	M.P.H.
1	18.96	18.96	224	63.2	156470	0	+9830	20.85	18.63
2	18.24	17.52	194	60.1	134150	+ 53680	—	22.32	17.59
3	21.48	20.22	167	41.1	55320	+109550	+	21.90	20.91
4	18.60	18.78	260	82.4	245280	— 14030	+	27.76	19.01
5	19.32	18.06	420	35.9	135870	+ 98200	+	30.58	19.01
6	7.62	7.62	112	67.9	184360	0	—	21.88	8.29
7	21.66	21.48	268	61.2	156580	+ 16190	+	22.89	21.63
8	21.12	19.50	178	40.3	57780	+137200	+	25.68	20.68
9	18.60	17.88	225	66.4	176840	+ 54760	+	27.80	17.59
10	14.28	15.18	227	80.2	240470	— 55280	—	21.98	16.22
11	26.16	26.52	383	202.7	342260	— 39540	+	39.18	26.50
12	26.34	26.34	383	164.5	277660	0	+	33.58	26.14
13	19.32	18.78	193	57.5	120240	+ 42900	—	21.69	18.45
14	21.30	21.12	312	164.3	266140	+ 15920	+	34.13	21.38
15	19.86	20.04	260	71.6	194000	— 14970	+	23.68	20.45
16	5.82	5.82	91.8	70.9	190650	0	—	22.67	6.84
17	14.28	13.38	142	56.4	109640	+ 51900	+	21.48	14.26
18	15.72	13.56	113	42.0	53860	+131870	+	24.52	14.82
19	15.90	15.00	164	52.7	106040	+ 57980	+	21.80	15.68
20	6.00	7.26	106	75.4	214790	— 34840	+	21.33	7.53
21	20.04	19.86	247	67.8	172010	+ 14970	+	24.67	20.45
22	16.98	17.52	220	78.3	224390	— 38840	+	24.49	16.36
23	26.52	25.26	290	150.0	190670	+136030	+	39.72	24.76
24	22.74	22.38	264	59.8	139180	+ 33870	+	22.93	23.23
25	13.02	12.66	140	54.0	108800	+ 19280	+	17.29	13.35
26	19.68	19.32	230	61.4	147760	+ 29270	+	23.43	19.60
27	19.68	19.32	243	65.2	168270	+ 29270	+	26.00	19.60
28	3.48	4.02	98.6	67.2	253450	— 8440	—	29.48	4.94
29	19.14	19.14	218	59.0	135510	0	+	18.22	19.20
30	20.04	20.04	264	76.7	220480	0	+	28.87	19.81
31	13.38	12.30	128	49.2	89600	+ 57830	+	19.72	12.98
32	33.90	34.08	573	194.7	396700	— 25510	+	45.30	33.60
33	14.28	14.28	164	51.4	107630	0	+	14.73	14.93
34	6.18	9.96	151	88.6	313150	—127200	—	22.08	8.92
35	20.22	20.40	243	63.8	158880	— 15240	+	19.24	20.23
36	21.48	23.10	370	199.1	382450	—150580	—	27.84	22.40
37	24.00	23.64	318	149.1	212320	+ 35760	—	29.87	24.76
38	21.56	21.56	266	61.6	154840	0	+9840	20.61	21.88
39	21.56	21.74	279	72.3	207240	— 16250	—	22.67	20.91
40	17.06	15.44	144	44.1	69260	+109780	+	23.64	16.08
41	13.82	15.26	189	71.8	189400	— 87310	+	14.01	14.93
42	8.06	5.18	75	54.7	118880	+ 79500	+	26.06	6.49
43	20.12	19.76	226	55.0	127340	+ 29930	+	20.92	19.40
44	18.86	18.32	223	53.5	128320	+ 41860	+	22.53	18.27
45	22.46	21.20	217	47.7	87850	+114700	+	26.58	22.14
46	22.10	21.74	300	68.0	187670	+ 32910	+	28.83	23.23
47	8.96	8.60	96	54.2	116760	+ 13180	+	17.49	8.44
48	20.66	20.30	272	66.5	182270	+ 30740	+	27.89	20.91
49	9.68	0.00	49.5	34.8	35960	+195370	+	30.18	6.51
50	13.64	11.66	120	46.5	82010	+104450	+	24.57	12.22

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UNIVERSITY OF ILLINOIS BULLETIN

ISSUED WEEKLY

VOL. XIV

NOVEMBER 6, 1916

NO. 10

[Entered as second-class matter Dec. 11, 1912, at the Post Office at Urbana, Ill., under the Act of Aug. 24, 1912]

A PRELIMINARY STUDY OF THE ALLOYS OF CHROMIUM, COPPER, AND NICKEL

BY
D. F. McFARLAND
AND
O. E. HARDER



BULLETIN NO. 93
ENGINEERING EXPERIMENT STATION

PUBLISHED BY THE UNIVERSITY OF ILLINOIS, URBANA

PRICE: THIRTY CENTS
EUROPEAN AGENT
CHAPMAN AND HALL, LTD, LONDON

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UNIVERSITY OF ILLINOIS

ENGINEERING EXPERIMENT STATION

BULLETIN No. 93

NOVEMBER 6, 1916

A PRELIMINARY STUDY OF THE ALLOYS OF CHROMIUM, COPPER, AND NICKEL

BY

D. F. McFARLAND

ASSISTANT PROFESSOR OF APPLIED CHEMISTRY,

AND

OSCAR E. HARDER

FELLOW IN CHEMISTRY

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A PRELIMINARY STUDY OF THE ALLOYS OF CHROMIUM, COPPER, AND NICKEL*

I. INTRODUCTION

1. *Purpose of the Investigation.*—The growing interest in special acid-resisting alloys and the many uses found for them has stimulated both the search for efficient materials of this nature and the study of the causes underlying their inertness. The alloys developed by Professor S. W. Parr for use in calorimeter construction have shown this quality of high resistance to corrosion to a marked degree. The almost perfect insolubility of these alloys in nitric and other acids seems to be conditioned upon a proper mixture of chromium, copper, and nickel, together with smaller quantities of such added metals as tungsten or molybdenum.†

These additions have so marked an effect in improving both the acid resisting properties and the casting qualities of the alloys that it has seemed desirable to study their effects more systematically in order that they may be used to the best advantage. The complexity of the mixtures used, however, has made the problem a very difficult one and has shown the necessity of first obtaining a more complete knowledge of the ternary alloys of chromium, copper, and nickel, and of the binary alloys of copper and nickel, copper and chromium, and chromium and nickel. With this information in hand it should be possible to understand better the effects produced by additions of a fourth metal.

In the work presented in this bulletin a preliminary survey has been made of the binary and ternary alloys and a somewhat systematic study made of their properties.

2. *Present State of Information.*—Previous investigators have studied in some detail the equilibrium conditions in the three binary systems. The freezing point curves for copper-nickel and chromium-nickel alloys have been fairly well established, but that for the chromium-copper series has been determined only in part, owing to the unusual difficulties attending the work. According to available information, these alloys belong to three different classes; copper-nickel

*The manuscript for this bulletin was originally prepared by Mr. Oscar E. Harder and submitted by him in partial fulfillment of the requirements for the degree of Doctor of Philosophy, University of Illinois. Since its presentation as a thesis, it has been reviewed and in certain particulars revised by Dr. D. F. McFarland.

†S. W. Parr, *Orig. Com. 8th Inter. Congr. Appl. Chem.* Vol. 2, p. 209, 1912. Also *Jour. Am. Chem. Soc.* Vol. 37, pp. 2515-2522, 1915.

forming a continuous series of solid solutions (or mixed crystals), chromium-nickel consisting of solid solutions with a minimum point which is said to be a eutectic point, and chromium-copper having two eutectic points with a wide range in which the metals do not alloy.

The alloys of copper and nickel are commercially important in the form of Monel metal (Cu 30 per cent, Ni 70 per cent), Constantan (Cu 60 per cent, Ni 40 per cent) and other resistance alloys, and nickel coin (Cu 75 per cent, Ni 25 per cent). Chromium-nickel alloys such as "Nichrome" and similar resistance alloys have important uses. Chromium-copper alloys of a limited range of composition are also obtainable on the market. The solubility of chromium in copper has been placed at from 0.5 to 10.0 per cent by various investigators, while the solubility of copper in chromium has been placed at less than 5.0 per cent.

3. *Plan of Work.*—It was originally intended that this investigation should include the following:

a. The preparation of samples of both the binary and ternary alloys which would represent all of the possible combinations with variations of 10 per cent of the different constituents.

b. Physical and mechanical examinations which would include color, appearance, specific gravity, hardness, tensile strength, reduction of area, elongation, and modulus of elasticity.

c. Corrosion tests in solutions of nitric acid, hydrochloric acid, sulfuric acid, sodium hydroxide, ammonium hydroxide, and sodium chloride, and in fatty acids.

d. Measurements of the relative electromotive forces of the alloys in contact with salt solutions.

e. Thermal analysis and heat treatment.

f. Microscopic examination.

This program has been carried out with the exception of the thermal analysis and heat treatment. Considerable time and effort have been expended on the attempt to work out the former, but the experimental difficulties have been so great that it has been necessary to reserve this analysis for further work and future report. It was thought best also to postpone the study of heat treatment until the equilibrium diagram could be obtained.

4. *Summary of Results and Conclusions.*—The results obtained together with the conclusions drawn from them may be summarized as follows:

a. Methods have been developed for making castings of alloys of chromium, copper, and nickel; and twenty-one binary and thirty ter-

nary alloys have been prepared. From this part of the work the following conditions have been drawn.

(1) Castings of chromium and copper containing as much as 13 per cent of chromium can be prepared by melting and pouring the metals at about 1600 degrees C.

(2) Chromium-copper alloys containing 6.08 per cent or more of chromium show a separation of chromium or of a chromium-rich constituent, if they are cooled slowly.

(3) If equal weights of chromium and copper are heated together to a temperature well above the melting point of chromium and slowly cooled, the alloy is not homogeneous, but consists of two layers; the lower rich in copper, and the upper rich in chromium.

(4) The addition of nickel to alloys of chromium and copper tends to prevent the separation of the chromium or chromium-rich constituent; when the amount of nickel is more than three times the amount of copper present, the alloys become practically homogeneous.

b. Physical and mechanical tests have been made with the following results:

(1) The specific gravity at 25 degrees C. of the alloys tested varies from 8.92 to 7.89 and decreases with an increase of chromium.

(2) The Brinell hardness number varies from that of pure copper to that of tool steel and increases with an increase of chromium.

(3) The modulus of elasticity of the sixteen alloys tested varies from less than 15,000,000 to more than 40,000,000 pounds per square inch. Generally it increases with an increase of chromium.

(4) The ultimate tensile strength of the eighteen alloys tested varies from less than 10,000 to more than 50,000 pounds per square inch.

(5) The reductions and elongations are small in all cases.

(6) The stress-deformation curves are similar to those of cast iron.

c. More than three hundred corrosion tests have been made. Results show that:

(1) The amount of corrosion is not proportional to the strength of the acid or base.

(2) A triangular system of plotting shows certain fairly well-defined areas which are highly resistant to corrosion.

(3) Not only the alloys in the region approximating the composition of the alloy developed by Professor Parr are highly non-corrodible, but others have shown equally good resistance to corrosion. In general the ternary alloys are less corroded than the binary, though there are some exceptions.

d. An attempt has been made to find some relation between the relative electromotive forces obtained by placing the alloys in contact with 4 Normal sodium chloride solution and their relative resistance to corrosion, but no such relation has been found from the experimental data obtained.

e. A microscopic study of the alloys has been made and the following agreements with earlier investigators have been found:

(1) The results agree with Voss' conclusions that chromium and nickel form a series of solid solutions (mixed crystals) over the range of 100 to 50 per cent of nickel and that they form a eutectic, or, as Guertler called it, a pseudoeutectic, containing about 42 per cent of nickel.

(2) The results on the copper nickel series agree with those of Guertler and Tammann in showing a continuous series of solid solutions.

(3) All nickel-rich alloys, both binary and ternary, show well-defined polyhedral crystals.

f. In general the results indicate that the alloys of chromium-copper-nickel which show possibilities of becoming of commercial importance are limited to certain rather well-defined ranges of composition. In Fig. 1 is shown diagrammatically what seems to be the most promising field for future investigation. In the nickel-rich corner of the diagram the alloys have very large polyhedral crystals and a coarse texture, and have usually developed blow-holes in casting. There is a possibility that the texture can be improved by the addition of a fourth metal as Professor Parr has done by the use of tungsten or molybdenum. The alloys containing large percentages of chromium have such high melting points and are so hard to prepare that, unless they find special and important applications, there is little chance of their being used commercially. Alloys containing large percentages of copper with chromium show such a marked segregation that they do not machine well and their mechanical properties are poor. The region which promises best from the mechanical and physical standpoint is also, as a general rule, highly resistant to corrosion in the various solutions tested.

II. PREPARATION OF THE ALLOYS.

5. *Materials.*—All the materials used in the preparation of the different alloys were of good quality. The chromium was secured from the Goldschmidt Thermit Company and was labeled 98.99 per cent

chromium. An analysis of one sample showed 98.2 per cent chromium with the remainder consisting largely of silica, slag, etc. The nickel was C. A. F. Kahlbaum's "Nickel in Würfeln" and an analysis showed that it contained 99.6 per cent of nickel, a small amount of iron, and only a trace of cobalt. Two different lots of copper were used, both of which were electrolytic; one purchased from the J. T. Baker Chemical Company and the other from Kahlbaum.

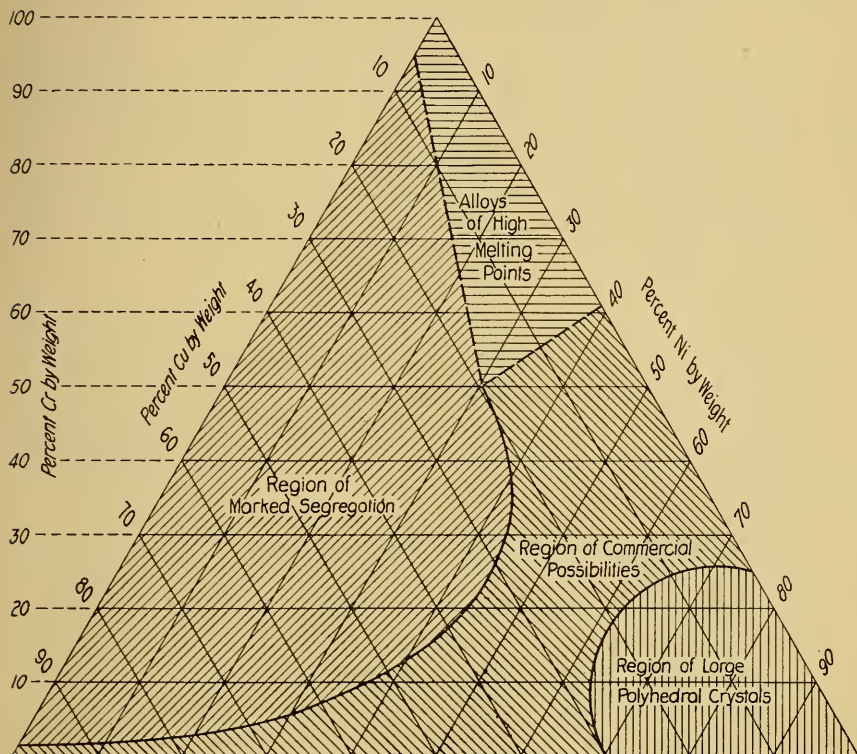


FIG. 1. DIAGRAM SHOWING ALLOYS HAVING POSSIBLE COMMERCIAL VALUES.

6. *Melting the Metals.*—The samples, with the exception of No. 1 to No. 6, inclusive, were melted in Crescent Safety crucibles in a Hoskins electric furnace of the carbon plate resistor type. These crucibles are sand crucibles covered with graphite. They withstood temperatures as high as 1600 degrees C., and in only one or two cases did they seem to be softened by that heat. It was not possible to use a crucible for more than one melt because of the corrosion of the sand lining. Samples No. 1 to No. 6, inclusive, were melted in a gas-fired

furnace in fire-clay crucibles. The metals were protected by a cover of powdered cryolite (NaAlF_4) which melted easily below the melting point of copper and effectively prevented oxidation of the chromium. It was not volatilized at the temperatures used and was the most satisfactory cover, although a number of other covers and fluxes were tried.

The charges of metals were of uniform size, 300 grams, in all cases. The furnace used from 25 to 30 kw. per hour and the time required for a melt varied from two to three hours.

7. *Casting the Samples.*—The molten metals were poured into asbestos-lined iron moulds $\frac{3}{4}$ inches in diameter and about 8 inches in length, which had been heated to a bright red temperature and packed in amorphous silica. By taking a reasonable amount of care it was possible to prevent contact of the molten alloy with the iron mould and to insure its easy removal from the mould when cold. The silica, being a good non-conductor of heat, allowed the casting to cool very slowly.

8. *Composition of Alloys.*—The intended composition of the different alloys is given in Table 1. The percentages are expressed in both atomic per cent and weight per cent. The weight per cent was calculated from the atomic per cent by means of the following formula in which *A*, *B*, and *C* represent respectively the three metals chromium, copper, and nickel.

$$\text{Weight percent } A = \frac{\text{At. \% } A \times \text{at. wt. } A \times 100}{\text{At. \% } A \times \text{at. wt. } A + \text{at. \% } B \times \text{at. wt. } B + \text{at. \% } C \times \text{at. wt. } C}.$$

In these calculations the values $\text{Cr} = 52.00$, $\text{Cu} = 63.57$, and $\text{Ni} = 58.68$ were used as the atomic weights. Table 1 shows also the weight per cent composition of the castings found by analysis.

9. *Analysis.*—Three methods of sampling were used. For some of the softer alloys the sample was obtained by making drillings; for the specimens which were turned to test pieces the turnings were used for analysis; in all other cases a piece was taken from one end of the specimen. The copper and nickel were determined electrolytically.* Chromium was determined by precipitating it as the hydroxide, and by igniting and weighing it as the oxide. In a few cases one of the constituents was determined by difference as indicated in Table 1.

10. *Changes in Composition.*—A comparison of the composition of the charge with the composition of the casting, as found by analysis,

*The preparation of the samples for analysis and the electrolytic determination of copper and nickel in all of the alloys were done by Sydney M. Hull.

TABLE 1
COMPOSITION OF ALLOYS

Number	Atomic Per Cent Composition			Weight Per Cent Composition			Weight Per Cent Composition by Analysis		
	Cu	Cr	Ni	Cu	Cr	Ni	Cu	Cr	Ni
1	100	...	...	100	...	...	...	...	...
2	90	...	10	90.71	...	9.29	90.84	...	9.06
3	80	...	20	81.27	...	18.73	81.07	...	18.76
4	70	...	30	71.65	...	28.35	71.16	...	28.46
5	60	...	40	61.90	...	38.10	61.63	...	38.25
6	50	*	50	52.00	...	48.00	48.96	...	49.90
7	40	...	60	41.94	...	58.06	69.13	...	30.59
8	30	...	70	31.70	...	68.30	41.14	...	58.27
9	20	...	80	21.31	...	78.69	20.65	...	79.35†
10	10	...	90	10.74	...	89.26	10.57	...	88.90
11	...	...	100	...	...	100.00	00.00	...	99.66
12	90	10	...	91.68	9.32	...	94.20	6.08	00.00
13	80	10	10	82.12	8.40	9.48	84.86	7.79	9.28
14	70	10	20	72.43	8.45	19.12	74.63	8.25	16.05
15	60	10	30	62.58	8.53	28.89	66.32	10.63	22.94
16	50	10	40	52.57	8.60	38.83	54.71	15.94	29.35
17	40	10	50	42.40	8.67	48.93	42.56	10.13	48.00
18	30	10	60	32.05	8.75	59.20	32.69	13.97	54.16
19	20	10	70	21.55	8.81	69.64	20.85	11.80	66.25
20	10	10	80	10.87	8.89	80.24	11.83	11.90	76.27†
21	...	10	90	...	8.96	91.04	00.00	19.37	78.99
22	80	20	...	83.02	16.98	...	87.93	13.15	00.00
23	70	20	10	73.22	17.12	9.66	80.58	10.56	9.38
24	60	20	20	63.28	17.25	19.47	56.30	14.56	29.24
25	50	20	30	53.16	17.40	29.44	66.92	13.62	19.20
26	40	20	40	42.88	17.54	39.58	44.08	19.30	36.34
27	30	20	50	32.41	17.69	49.90	36.70	15.99	47.31†
28	20	20	60	21.80	17.83	60.37	22.20	19.86	57.36
29	10	20	70	10.99	17.98	71.03	10.88	19.64	68.62
30	...	20	80	...	18.14	81.86	00.00	21.52	76.95
31	70	30	...	74.05	25.95	...	89.82	9.89	00.00
32	60	30	10	63.99	26.17	9.84	73.63	17.66	8.55
33	50	30	20	53.77	26.38	19.85	59.62	22.00	19.48
34	40	30	30	43.37	26.61	30.02	45.70	25.10	29.52
35	30	30	40	32.79	26.84	40.37	33.76	29.46	36.78†
36	20	30	50	22.05	27.06	50.89	22.58	28.10	48.42
37	10	30	60	11.12	27.29	61.59	10.90	29.70	58.12
38	...	30	70	...	27.52	72.48	00.00	28.44	71.56†
39	60	40	...	64.71	35.29	...	Not analyzed		
40	50	40	10	54.38	35.58	10.04	70.57	19.93	8.99
41	40	40	20	43.86	35.89	20.25	54.16	31.63†	14.21
42	30	40	30	33.17	36.20	30.63	33.60	38.16	26.78
43	20	40	40	22.31	36.50	41.19	22.68	41.32	34.60
44	10	40	50	11.25	36.82	51.93	11.02	43.30	46.46
45	...	40	60	...	37.14	62.86	00.00	44.93	56.55
46	50	50	...	55.20	44.80	...	Not prepared		
47	40	50	10	44.38	45.37	10.25	Not prepared		
48	30	50	20	33.57	45.76	20.67	28.42	54.92	17.12
49	20	50	30	22.53	46.16	31.26	24.12	47.54	26.28
50	10	50	40	11.39	46.56	42.05	...	...	...
51	...	50	50	...	46.96	53.04	00.00	57.40	41.66
52	40	60	...	44.89	55.11	...	...	...	...
53	30	60	10	33.96	55.59	10.45	...	...	...
54	20	60	20	22.85	56.06	21.09	...	...	...
55	10	60	30	11.53	56.56	31.91	Not analyzed		
56	...	60	40	...	57.07	42.93	Not analyzed		
57	30	70	...	34.37	65.63	...	...	...	...
58	20	70	10	23.13	66.21	11.56	...	...	...
59	10	70	20	11.66	66.80	21.54	...	...	...
60	...	70	30	...	67.40	32.60	Not analyzed		
61	20	80	...	23.41	76.59	...	...	...	...
62	10	80	10	11.81	77.29	10.90	...	...	...
63	...	80	20	...	78.00	22.00	Not analyzed		
64	10	90	...	11.96	88.04	...	...	...	...
65	...	90	10	...	88.86	11.14	Not analyzed		
66	...	100	...	...	100.00	...	...	98.21	...

*No. 6 was made up to be 50 per cent copper and 50 per cent nickel by weight.

†Determined by difference.

shows that there was a remarkable loss in chromium in the chromium-copper alloys. The highest chromium content found was in No. 22 which showed 13.15 per cent, whereas it should have shown about 17 per cent. In No. 31 the chromium should have been about 26 per cent, but analysis showed only 9.89 per cent. All attempts to prepare alloys of chromium and copper containing higher percentages of chromium failed. Even in the case of Alloy No. 12, which was slowly cooled and which contained only 6.08 per cent of chromium, there is a separation of pure chromium or of a chromium-rich constituent. (See Fig. 25.)

It has been stated by Hindrichs that at higher temperatures chromium and copper form an emulsion in which the chromium may be in a finely divided condition, and that on slow cooling the chromium collects in larger particles. It seems more probable, however, that chromium should be more soluble in copper at higher temperatures than at the melting point. In either case rapid cooling would tend to prevent such a separation and to produce a more homogeneous structure. Evidence to this effect is shown by the fact that the alloy containing 10 per cent of chromium and 90 per cent of copper which is offered for sale by the Goldschmidt Thermit Company is almost homogeneous. This alloy is said to be made by an aluminothermic method. The aluminothermic reaction produces a very high temperature, and if, as appears to be the case, the alloy is cooled suddenly, there would not be sufficient time for the separation of the chromium or chromium-rich constituent.

In the case of the other alloys there were losses of different constituents depending upon a number of conditions. In the ternary alloys rich in copper there was generally a loss in chromium, probably due to the insolubility of the chromium in the other metals present. In the ternary alloys rich in nickel and in the binary alloys of chromium and nickel, there was generally a loss of nickel, but that may be accounted for by the fact that in making up the charges precaution was taken to protect the chromium by putting it in the bottom of the crucible, thus the nickel was left more exposed to oxidation. The changes in composition may be studied in Table 1. It is certain that in making a series of alloys by such a method as has been outlined, it is not safe to assume that the composition of the alloy obtained will be the same as that of the charge melted.

11. *Alloy of 50 Per Cent Cr and 50 Per Cent Cu.*—In order to determine whether or not a homogeneous alloy containing equal parts

of chromium and copper could be prepared, the following experiments were conducted.

A charge of 30 grams of chromium and 30 grams of copper was heated in an electric furnace. A little copper was put in the bottom of the crucible, the chromium was added, and then the remainder of the copper. The charge was covered with cryolite. It was heated well above the melting point of chromium and kept at that temperature for at least fifteen minutes. The current was then shut off and the furnace allowed to cool slowly. An examination showed that the chromium had been melted. The copper which had been put on top of the charge was found at the bottom of the melt, and a fairly well defined line of separation between the copper-rich and the chromium-rich parts of the melt could be seen. (See Figs. 2 and 3.) The above experiment was later repeated with similar results. By this method it has been possible to get a division of the melt into two fairly well defined layers. From a microscopic examination (Figs. 2 and 3) it was evident that the lower layer contained some chromium, and it appeared that the upper layer contained some copper. Hindrichs stated that he was not able to get the two metals to separate into two sharply defined layers.



FIG. 2

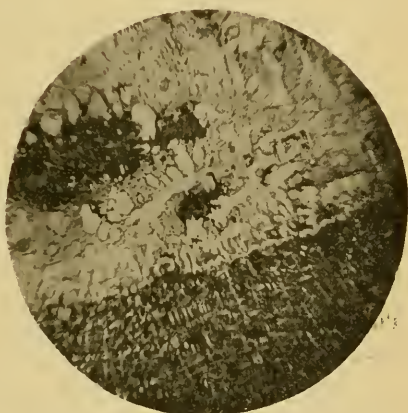


FIG. 3

MICROPHOTOGRAPHS SHOWING SEPARATION OF COPPER FROM CHROMIUM IN A 50 PER CENT CU, 50 PER CENT CR. ALLOY. ETCHED IN 1 PER CENT FeCl_3 IN 1:1 HCl AND THEN STAINED IN I IN ALCOHOL. THE DARK PORTION IS THE COPPER-RICH CONSTITUENT. MAGNIFIED 30 DIAMETERS.

III. PHYSICAL AND MECHANICAL PROPERTIES

12. *Color and Appearance.*—The colors of the different alloys depend upon the amount of copper as compared with the sum of the amounts of chromium and nickel present. The chromium and nickel colors are more persistent than the copper color, and an alloy containing 50 per cent of copper and 50 per cent of nickel, or of chromium and nickel, has the color of nickel and does not show any color of copper. The alloys near the pure copper corner of the diagram (see Fig. 4) show the greatest tendency to tarnish when exposed to the air in the laboratory. On the other hand, the alloys near the nickel corner of the diagram are more porous and more likely to contain blowholes.

13. *Method of Plotting Diagrams.*—The following method has been adopted for plotting the relation between the compositions and the various properties of the alloys as shown in Fig. 4.

The three metals, chromium, copper, and nickel, are represented by the corners of the equilateral triangle. The compositions of the different alloys, as found by analysis, have been represented by the centers of the circles so that they can be read directly from the diagram. The number of the alloy has been placed inside of the circle. All of the binary alloys fall on the sides of the triangle and all of the ternary alloys fall within the triangle. One example will serve to show the method of reading the composition of the different alloys from their positions on the diagram. No. 28 is approximately on the line marked 20 per cent chromium which runs parallel to the copper-nickel side of the triangle and between the lines marked 60 and 50 per cent of nickel which run parallel to the chromium-copper side of triangle. Therefore it can be estimated that the alloy contains approximately 57 per cent of nickel. About 23 per cent is left for copper. The same result can be read from the distance of the center of the circle to the line marked 20 per cent copper which runs parallel to the chromium-nickel side of the triangle. By this method it is possible to read directly the composition of any alloy which has been plotted on the diagram. The alloys which show the copper color are Nos. 2, 3, 4, 5, 12, 13, 14, 22, 23, 31, 32, 40, and 48. Their compositions can be read from Fig. 4 or can be found in Table 1. Table 2 shows in a brief way the kinds of castings which were obtained for the different alloys.

14. *Specific Gravity.*—The specific gravities of the different alloys, as cast, are given in Table 2, and the same results are shown in Fig. 4. In the diagram the specific gravity of the alloy has been written above the circle representing the composition of the alloy.

The alloys were weighed in air and in distilled water at 25 degrees C., and the specific gravities were determined from the formula:

$$\text{Sp. gr.} = \frac{\text{Wt. in air at } t^{\circ}\text{C.} \times \text{sp. gr. H}_2\text{O at } t^{\circ}\text{C.}}{\text{Loss of wt. in H}_2\text{O at } t^{\circ}\text{C.}}$$

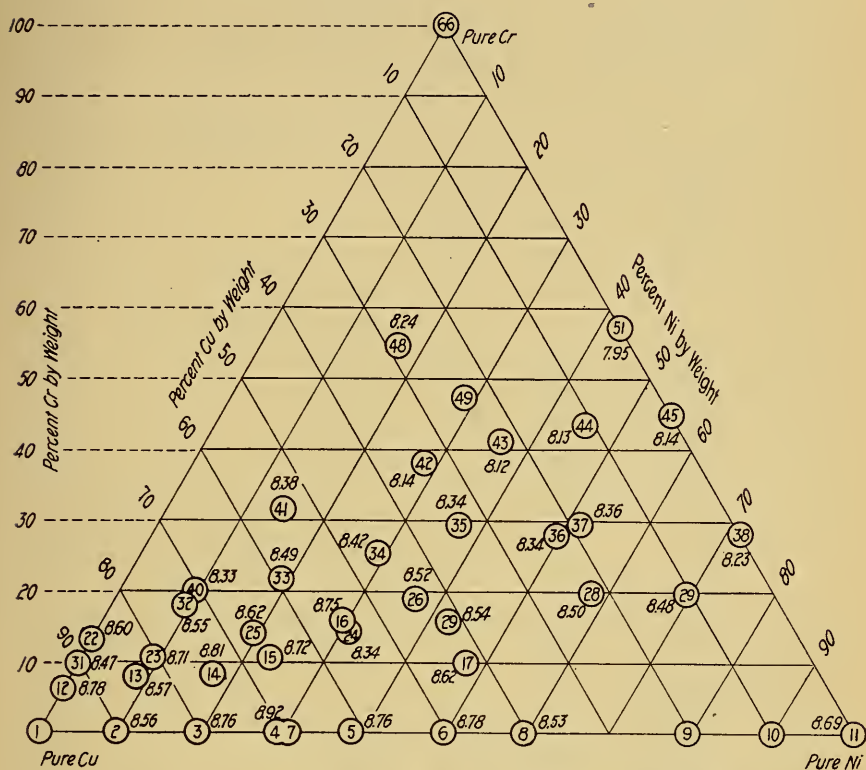


FIG. 4. DIAGRAM SHOWING COMPOSITION OF ALLOYS AND SPECIFIC GRAVITIES AT 25 DEGREES C.

If the alloys containing a constant amount of chromium with varying amounts of copper and nickel are considered, it will be seen that there is little variation in the specific gravity. On the other hand, if the alloys containing a constant amount of copper with varying amounts of chromium and nickel are considered, it will be seen that the specific gravity decreases as the percentage of chromium increases. Similar results may be obtained if the alloys containing constant amounts of nickel and varying amounts of chromium and copper are examined. If the specific gravities are plotted as ordinates and the percentages of chromium as abscissae, fairly regular curves will be obtained. When

TABLE 2
HARDNESS, SPECIFIC GRAVITY, AND KIND OF CASTING

Number	Diameter of Impression	Brinell Hardness Number	Specific Gravity at 25° C.	Kind of Casting
1	...	...	...	Good.
2	Broke in test.	...	8.56	Good.
3	Cracked in test.	...	8.76	Medium.
4	Cracked in test.	...	8.92	Blowholes in places.
5	Specimen too small.	...	8.76	Blowholes in places.
6	Specimen too small.	...	8.78	Good.
7	7.20	Below 68	8.53	Good.
8	...	...	...	Casting had blowholes.
9	...	...	...	Blowholes all through.
10	...	...	...	Blowholes in places.
11	Broke in test. (5.4 mm. thick)	...	8.69	Large crystals and small blowholes.
12	7.20	Below 68	8.78	Good.
13	6.80	86	8.57	Sound.
14	5.80	103	8.81	Sound.
15	5.00	143	8.72	Sound.
16	4.73	161	8.75	Small, but sound.
17	4.70	163	8.62	Fair.
18	...	...	...	Small blowholes.
19	...	...	...	Fair, except for blowholes.
20	...	...	...	Fair, except for blowholes.
21	...	...	...	Large blowholes.
22	6.90	69	8.60	Sound throughout.
23	6.40	82	8.71	Good.
24	6.20	89	*8.34	Good.
25	5.90	99	8.62	Good.
26	4.80	156	8.52	Very good.
27	4.50	175	8.54	Sound.
28	4.50	179	8.50	Sound.
29	5.30	126	8.48	Sound.
	(Specimen small)			
30	...	...	...	Large blowholes.
31	7.30	68	8.47	Medium.
32	5.46	117	8.55	Small, but sound.
33	5.50	116	8.49	Excellent. There was some segregation of Cr at top of casting.
34	5.30	126	8.42	Good.
35	4.92	148	8.34	Excellent.
36	4.60	170	8.34	Excellent.
37	4.70	163	8.36	Excellent.
38	4.48	181	8.23	Good.
39	Not prepared.	...	...	...
40	5.40	121	8.33	Small, but sound.
41	5.50	116	8.38	Good.
42	5.20	131	8.14	Excellent.
43	4.80	156	8.12	Excellent.
44	4.70	163	8.13	Excellent.
45	4.93	147(?)	8.14	Good.
	(Cracked in test.)			
46	Not prepared.	...	...	...
47	Not prepared.	...	...	...
48	5.70	107	8.24	Excellent.
49	...	...	...	Sound.
50	4.25	202	7.89	Good.
51	4.50	179	7.95	Good.
55	4.00	228	...	Sound.
56	4.50	179(?)	...	Sound.
	(Specimen too small)			
60	...	...	...	Sound.
63	...	...	...	Sound.
65	...	...	...	Sound.

*This value is in error because of a concealed blowhole.

the exact composition of the alloy is known, it is possible to get some idea of the relative porosity of the specimen by a study of the specific gravity.

15. *Brinell Hardness Number.*—The hardness measurements were made in the Materials Testing Laboratory of the Department of Theoretical and Applied Mechanics of the University of Illinois by using a Brinell instrument with a 3,000 kilogram load and a ball of 10 mm. diameter, the pressure being applied for 15 seconds. The diameters of the impressions were carefully measured, and the hardness numbers corresponding to those diameters were found in a table supplied by the makers of the instrument. The pieces tested had been ground smooth and were from 4 to 8 mm. thick and about 20 mm. in diameter. Only one test was made on each piece, and if it was noticed that the piece had bulged or cracked from the pressure, that fact was recorded or the results were rejected.

Specimen No. 11 requires special mention. It was a sample of pure nickel which had been melted in a magnesia crucible and slowly cooled. It had very large crystals (see microphotograph of Alloy No. 11, Fig. 24), and when a piece 5.4 mm. thick and 22 mm. in diameter was tested, it broke at a load of about 2,000 kilograms.

A comparison with the values which are given in Table 3* shows that some of the specimens were especially hard.

TABLE 3
HARDNESS TESTS BY CANADA DEPARTMENT OF MINES

Material	Date	Load	Brinell Hardness
Copper, rolled sheet, unannealed	Jan. 1913	1,000 lbs.	65.6
" " " "	" 1914	1,000 "	67.4
" " " "	" 1914	3,500 "	75.0
" " " "	" 1914	3,500 "	81.9
Swedish iron	Jan. 1913	3,500 "	90.7
" "	" 1914	1,000 "	68.6
" "	" 1914	3,500 "	75.2
Wrought iron	Jan. 1913	3,500 "	92.0
" "	" 1914	1,000 "	83.1
" "	" 1914	3,500 "	100.2
Cast iron	Jan. 1913	3,500 "	97.8
" "	" 1914	1,000 "	84.4
" "	" 1914	3,500 "	104.5
Mild steel	Jan. 1913	3,500 "	109.9
" " cold rolled shafting	" 1914	3,500 "	126.2
Tool steel	Jan. 1913	3,500 "	153.8
" " "Crescent"	" 1914	3,500 "	130.2
Spring steel	Jan. 1913	3,500 "	160.3
" "	" 1914	3,500 "	178.0
Tool steel self-hardening	Jan. 1913	3,500 "	180.0
" " " "	" 1914	3,500 "	162.1
Tool steel, self-hardening	" 1914	3,500 "	240.0
" " "Rex" (after hardening)	" 1914	3,500 "	240.0
Tool steel, self-hardening, from work shop	Jan. 1914	3,500 "	259.0
(School of Mines)			
Cobalt		3,500 "	124.0

*Canada Department of Mines, Report No. 309, Part II, p. 9, by Kalmus and Harper.

It is evident that some of these alloys are as hard as tool steel, and No. 55 approaches the hardness of self-hardening tool steel after hardening. It is likely that some of the alloys in the series have still greater hardness than the one referred to since some of the others contain more chromium.

These specimens were extremely difficult to work. A saw blade was ruined on some of them without cutting more than 1 mm. deep. It has been found that by keeping the saw blades moist with turpentine they will last somewhat longer.

The results obtained by the hardness tests are shown in Table 2.

16. *Tensile Strength and Stress-Deformation Tests.*—Eighteen specimens of the alloys of various compositions were turned down to test pieces and their ultimate tensile strengths determined. In all except two cases their elongations under stresses were measured.

Test Pieces.—The test pieces which were from 3 to 4 inches in total length were turned to a diameter of approximately 0.300 inches for a length of 2.0 inches, and were threaded at the ends so that they could be screwed into half-inch grips. It was necessary to make the measurements on these short test pieces instead of the usual 8-inch specimens, because the original castings were only 4 to 5 inches long and pieces had been cut off for microscopic examination, for corrosion tests, etc. It is likely that some of the irregularities observed in the values obtained for the modulus of elasticity and for the ultimate tensile strength should be attributed to the use of the short test pieces.

Testing Machine and Method of Loading.—The testing machine on which the tests were made was an Olsen Universal Screw-Power Testing Machine of 10,000 pounds capacity. The loading was by hand and was slow except in the case of specimens No. 17 and No. 28 when the loading was done with the motor. The loading was continuous and not repeated.

Extensometer.—A Ewing extensometer having a gage of 1.25 inches was used in measuring the elongations. The instrument was sensitive and the elongations could be read accurately to 0.00008 inches and estimated to 0.000008 inches. The initial or zero extensometer reading was taken with a small load on the machine. After a satisfactory number of readings had been made the extensometer was removed and the load increased until the specimen broke.

Calculation of Modulus of Elasticity.—Curves were plotted with stresses as ordinates and elongations as abscissae. Tangents to the curves were drawn and extended to a point which corresponded to an elongation of 0.001 inch on a length of 1.0 inch, and the stress cor-

responding to that elongation was read from the cross-ruled paper. This stress minus the initial stress for zero elongation gave the modulus of elasticity. The values are given in round numbers.

The results are shown in Table 4, and the specimens have been tabulated so that the percentage of chromium increases from the top to the bottom of the column.

TABLE 4
COMPOSITION, MODULUS OF ELASTICITY, AND ULTIMATE TENSILE STRENGTH

Number	Composition of the Alloy as Found by Analysis			Modulus of Elasticity in lb. per sq. in.	Ultimate Tensile Strength in lb. per sq. in.
	Cr	Cu	Ni		
14	8.25	74.63	16.05	18,000,000	19,978
17	10.13	42.56	48.00		23,833
25	13.62	66.92	19.20	7,000,000	14,880*
24	14.56	56.30	29.24	20,000,000	17,521
26	19.30	44.08	36.34	12,000,000	14,802
28	19.64	22.20	57.36		56,255
29	19.64	10.88	68.62	7,000,000	19,846†
33	22.00	59.26	19.48	17,600,000	22,650
34	25.10	45.70	28.52	20,000,000	25,768
36	28.10	22.58	48.42	23,700,000	32,654 plus
38	28.44		71.56	15,800,000	36,958
35	29.46	33.76	36.78	16,300,000	38,734
37	29.70	10.90	58.12	22,000,000	29,630
42	38.16	33.60	26.78	26,400,000	7,449
44	43.30	11.02	46.46	16,300,000	29,587
49	47.54	24.12	26.28	43,200,000	22,336 plus
48	54.92	28.42	17.12	57,000,000	33,842†
51	57.40		41.66	48,000,000	37,200

*Showed a flaw at the fracture.

†The threads were not straight and these values are considered less reliable than the others.

The following is given as a typical set of data. The complete data for each specimen is not included.

ALLOY No. 35

DIAMETER 0.305 IN. CROSS SECTION, 0.07306 sq. in.

Extensometer Readings	Elongation in .0008 in.	Load in Pounds	Unit Load	Remarks
3.00	0.0000	150	2,053	
3.30	0.30	380	5,201	
3.63	0.63	625	8,554	
3.93	0.93	894	12,236	
4.23	1.23	1140	15,603	
4.55	1.55	1385	18,956	
4.83	1.83	1625	22,241	
5.18	2.18	1790	24,500	
5.60	2.60	2060	27,374	
Ultimate		2830	38,734	

Reduction and Elongation.—The per cent of reduction of area was quite small in all cases, in fact, it was so slight that measurements with a micrometer would not give appreciable decreases in diameters. The curves in Fig. 5 show that the elongations were small. These alloys

do not show well-defined yield points or elastic limits but a gradual flattening of the stress deformation curves until the ultimate is reached. The curves are similar to those for cast iron.

Ultimate Strength.—The ultimate tensile strength varied from 7,449 to 56,255 pounds per square inch cross-section. The binary alloy of chromium and nickel had more ultimate tensile strength than the ternary alloy in which 10 per cent of the nickel had been replaced by copper. (See Nos. 38 and 37.) It does not seem possible, however, to draw any general conclusions from these results as to which composition will have the greatest tensile strength. The remarkably low ultimate strength for No. 42 is probably due to the fact that it contains a large amount of chromium (38.16 per cent) and at the same time a large amount of copper (33.60 per cent), which produces an unstable condition because it has not been possible to prepare binary alloys of copper and chromium containing more than about 13 per cent of chromium.

Modulus of Elasticity.—Although there are some irregularities in the values obtained, it seems that the modulus of elasticity increases with an increase in chromium content. The values obtained vary from 15,000,000 to more than 40,000,000, or by from $\frac{1}{2}$ to $1\frac{1}{2}$ times the value for iron and steel.

A general idea of the properties of these alloys may be obtained from the stress-deformation curves which are given in Fig. 5.

IV. CORROSION TESTS

17. *Reason for Tests.*—In order to determine the power of the different alloys to resist corrosion, over three hundred tests were made. It was hoped that such a series of tests would show that some of these alloys were highly resistant to corrosion and suited for industrial purposes requiring that property.

18. *Materials.*—The corroding reagents used were normal solutions of nitric acid, hydrochloric acid, sulfuric acid, sodium hydroxide, ammonium hydroxide, sodium chloride, and molten fatty acids. The normal solutions had been carefully standardized and were known to be reasonably accurate. The fatty acids were secured from Swift & Co. of Chicago and were listed as double distilled. An analysis showed that they had a mean molecular weight of 273, an iodine value of 72, and consisted of a mixture of fatty acids. The alloy specimens used in the tests were usually about 5 mm. thick and about 15 to 20 mm. in diameter. They were ground and then polished with

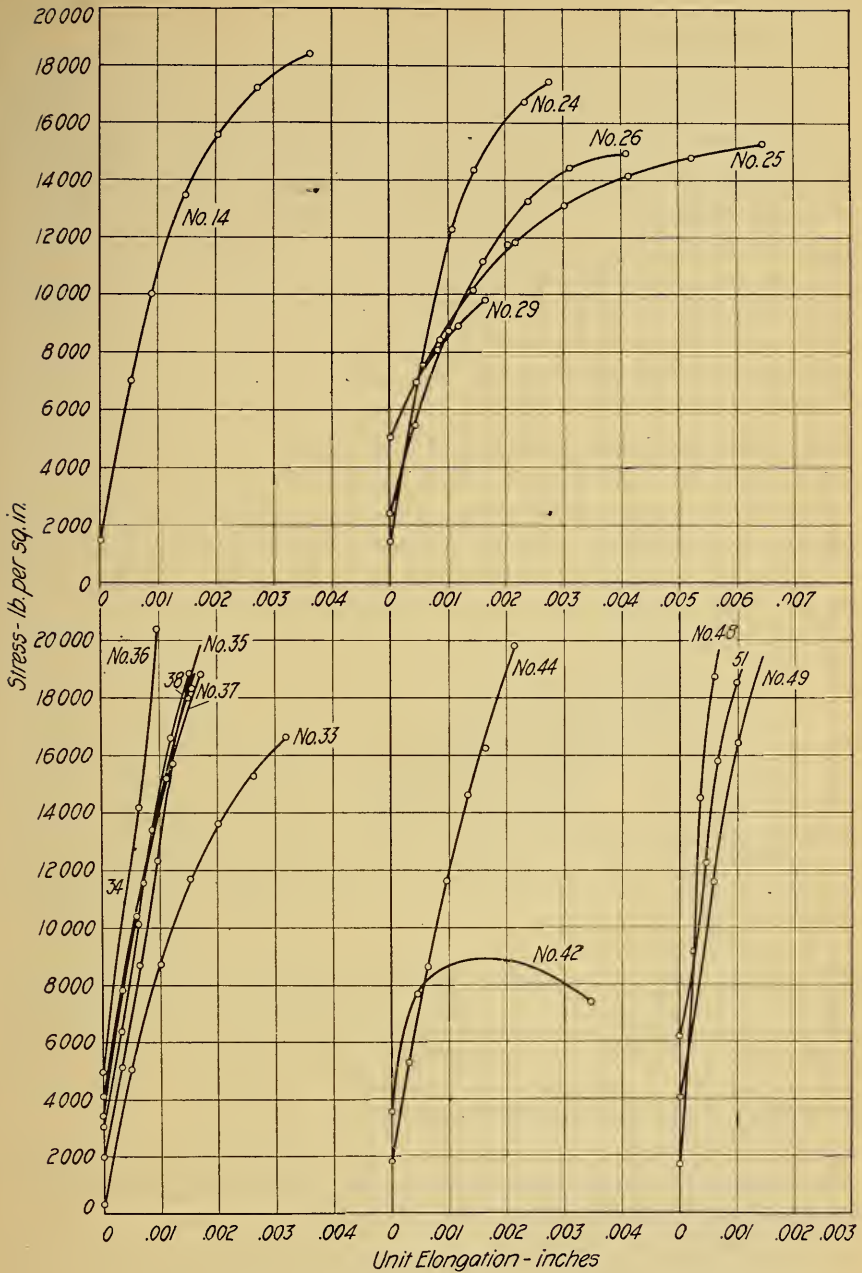


FIG. 5. STRESS DEFORMATION CURVES FOR CR-CU-NI ALLOYS.

No. 000 Hubert Emery paper. The three metals were included in the tests with the alloys. Fifty different samples were tested.

19. *Methods.*—There does not seem to be any well-established method for making corrosion tests. Different investigators have used a variety of methods. For the work in hand, the scheme used has been reasonably satisfactory and it will be described somewhat in detail.

Solutions.—In the case of the normal solutions mentioned under Section 18, 400 cc. were used for each test. No attempt was made to remove any dissolved gases, such as carbon dioxide. The solutions were contained in beakers of such size that they were filled almost to the top. Of course, the solutions changed slightly in concentration because of evaporation and of corrosion of the specimens, but that could not be avoided. The loss by evaporation was made very small by covering the beakers with watch glasses. With such a large volume of solution the change in concentration was not important in most cases. If it was observed that a specimen was dissolving very rapidly, it was removed and from the amount which had been corroded a calculation was made to find how much would have corroded in one week. One hundred grams of the fatty acids were taken for each of the tests that were made at 105 degrees C. and 200 grams for those made at 85 degrees C.

Suspending the Specimens.—Cotton thread was used to suspend the specimens in the alkaline solutions and wool thread was used in the acid solutions. The specimens were completely immersed in the liquid so that no part was in contact with the air. In some cases there was a tendency for the solutions to creep out of the beakers by way of the threads, but that was prevented by putting a little paraffin on the thread.

Temperature.—The corrosion tests were made at the temperature of the laboratory. In order to be able to approximate the mean temperature, a maximum and minimum thermometer was kept on the desk where the tests were being made and readings were taken from time to time. A calculation showed that the mean temperature for the whole period of time was 20 degrees C.

Time.—It was intended to leave the specimens in the solutions for one week, but in some cases it was necessary to remove them before the end of that period.

Calculations.—The amount of surface exposed to corrosion was determined by measuring the thickness and the diameter of the piece to the nearest 0.1 mm. and then calculating the total surface. Unless there was known to have been a big loss in corrosion, the surface was

not recalculated when the piece was used for a second test. The results have been calculated to loss in weight per week per square inch of surface exposed and have been expressed in milligrams.

Accuracy of the Method.—The accuracy of the method has not been all that might be desired. The specimens were weighed before and after the tests and the loss in weight was taken as the loss due to

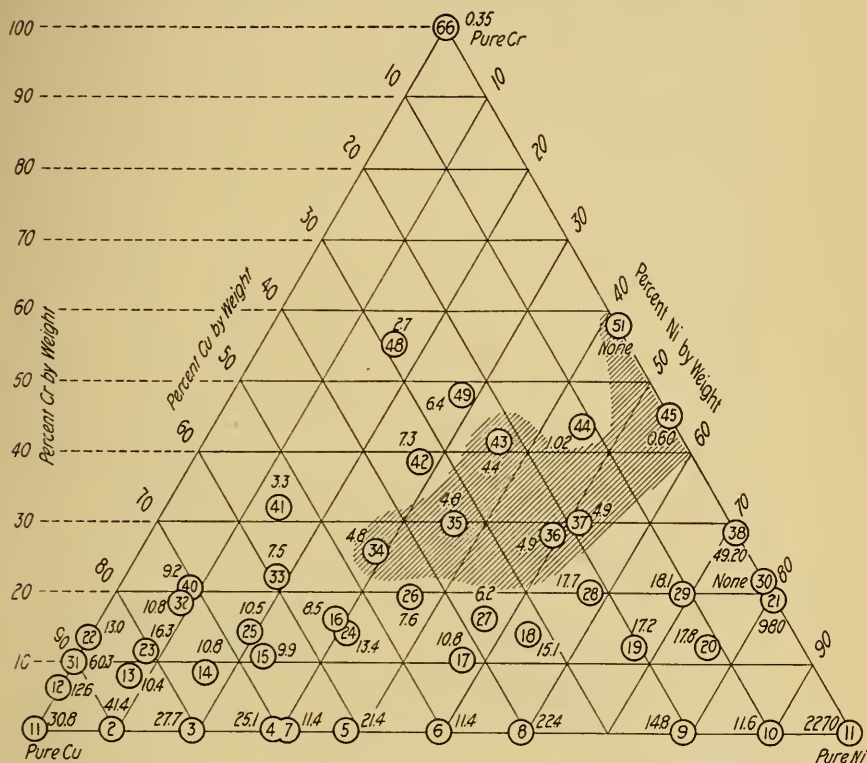


FIG. 6. DIAGRAM SHOWING COMPOSITION OF ALLOYS AND CORROSION IN NORMAL NITRIC ACID SOLUTION.

corrosion. In a few cases a gain in weight was observed. It was hard to get all of the salts, etc. out of some of the porous pieces, but by boiling in distilled water and drying at 100 degrees C. most of that trouble was avoided. While some of the results show irregularities, as a whole they may be considered reasonably accurate and reliable.

20. *Results.*—The results of the corrosion tests are shown on triangular diagrams as explained previously (p. 14). In each diagram the loss in weight in milligrams per square inch per week for each

alloy tested is shown above the circle, the center of which represents the composition of the alloy as determined by analysis.

Complete data of the tests with normal nitric acid are given in Table 5, and by comparison of this table with the corresponding diagram (Fig. 6), the method of plotting can be readily understood. Such detailed data has been omitted for the tests with other solvents, the results being shown only in the graphical form. Areas of minimum corrosion are designated on the diagrams by shading.

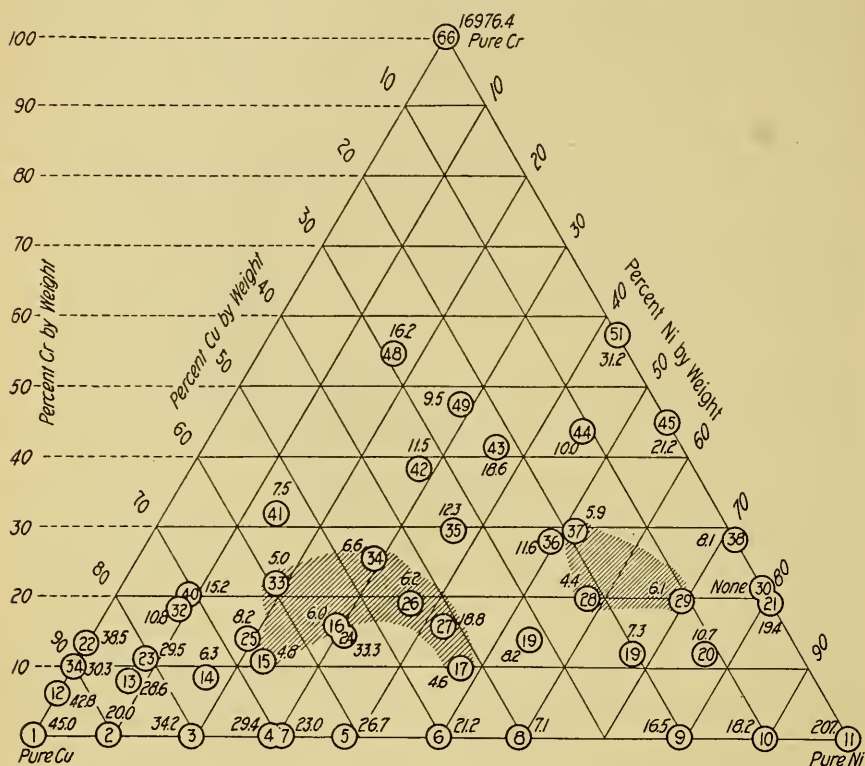


FIG. 7. DIAGRAM SHOWING COMPOSITION OF ALLOYS AND CORROSION IN NORMAL HYDROCHLORIC ACID SOLUTION.

Nitric Acid.—Table 5 and Fig. 6 show the results obtained with a normal solution of nitric acid. Nickel, especially, is attacked, and alloys with high percentages of this metal show great corrosion. In the chromium-nickel alloys there is a gradual decrease in the loss by corrosion as the per cent of chromium increases, with an abrupt drop at Alloy No. 45. There is also a marked change in the micro-structure

TABLE 5
CORROSION IN NORMAL NITRIC ACID

Number	Weight Before	Weight After	Loss in weight mg.	Surface		Time in solution hrs.	Loss in wt. in mg. per wk.	
				sq. in.	sq. cm.		sq. in.	sq. cm.
1	5.5125	5.4945	18.0	0.71	4.58	138	30.80	4.78
2	8.6232	8.5882	35.0	1.03	6.67	138	41.40	6.42
3	5.6478	5.6323	15.5	0.68	4.41	138	27.70	4.30
4	4.5331	4.5205	12.6	0.61	3.95	138	25.10	3.38
5	4.5007	4.4898	10.9	0.64	4.12	138	21.40	3.38
6	4.5918	4.5859	5.9	0.63	4.04	138	11.40	1.77
7	16.9032	16.8890	14.2	1.52	9.83	138	11.40	1.77
8	11.3342	11.3092	25.0	1.36	8.77	138	22.40	3.47
9	9.8259	9.8086	14.5	1.28	8.27	138	14.80	2.13
10	10.0842	10.0724	11.8	1.24	7.98	138	11.60	1.80
11	9.8700	9.4410	429.0	1.32	8.49	24	2270.00	352.00
12	11.7920	11.7783	13.7	1.33	8.58	137	12.60	1.98
13	13.1763	13.1647	11.6	1.35	8.69	137	10.40	1.62
14	10.0412	10.0310	10.2	1.16	7.47	137	10.80	1.67
15	8.4206	8.4123	8.3	1.03	6.66	137	9.90	1.53
16	7.3881	7.3807	7.4	1.07	6.94	137	8.50	1.32
17	12.8800	12.8684	11.6	1.32	8.51	137	10.80	1.67
18	10.5626	10.5474	15.2	1.24	7.98	137	15.10	2.33
19	14.3148	14.2931	21.7	1.54	9.92	137	17.20	2.67
20	10.6661	10.6490	17.1	1.18	7.61	137	17.80	2.76
21	13.2433	12.8326	410.7	1.47	9.43	48	980.00	152.00
22	9.0392	9.0390	10.2	1.07	6.91	122	13.10	2.03
23	9.1265	9.1151	11.4	1.03	6.67	122	16.30	2.22
24	10.5944	10.5821	12.3	1.26	8.10	122	13.40	2.08
25	11.1470	11.1376	9.4	1.23	7.92	122	10.50	1.63
26	13.1489	13.1412	7.7	1.39	8.95	122	7.60	1.18
27	9.7288	9.7222	6.6	1.08	7.00	168	6.20	0.95
28	9.8098	9.7948	15.0	1.17	7.56	122	17.70	2.74
29	8.0788	8.0655	13.3	1.03	6.65	122	18.10	2.80
30	Too many holes.							
31	11.1653	11.0955	69.8	1.16	7.46	168	60.30	9.35
32	12.1926	12.1777	14.9	1.38	8.92	168	10.80	1.67
33	9.7367	9.7270	9.7	1.29	8.33	168	7.50	1.16
34	11.6059	11.5994	6.5	1.37	8.82	168	4.80	0.74
35	8.8904	8.8847	5.7	1.18	7.62	168	4.80	0.74
36	9.6290	9.6241	5.9	1.20	7.73	168	4.90	0.76
37	10.8796	10.8738	5.8	1.31	8.43	168	4.90	0.76
38	9.2554	9.0129	242.5	1.18	7.62	70	492.00	76.00
39	Not prepared.							
40	13.7427	13.7295	13.2	1.45	9.33	168	9.20	1.43
41	11.5077	11.4963	11.4	1.37	8.83	168	8.30	1.29
42	10.1818	10.1725	9.3	1.28	8.23	168	7.30	1.13
43	11.5401	11.5341	6.0	1.35	8.70	168	4.40	0.69
44	11.8160	11.8146	1.4	1.37	8.81	168	1.02	0.16
45	10.5332	10.5325	0.7	1.16	7.46	168	0.60	0.09
46	Not prepared.							
47	Not prepared.							
48	10.7279	10.7154	12.5	1.29	8.32	168	9.70	1.50
49	9.1910	9.1854	5.6	1.21	7.81	122	6.40	0.99
50	10.4293	10.4198	9.5	1.28	8.23	168	7.40	1.15
51	12.4458	12.4458	0.0	1.42	9.19	168	0.00	0.00
55	9.7665	9.7559	10.6	1.38	8.89	168	7.70	1.19
56	8.4517	8.4513	0.4	1.14	7.33	168	0.35	0.05
66	2.0776	2.0774	0.2	0.57	3.66	168	0.35	0.05

of the alloys in the same region as may be seen by comparing microphotographs of Alloys No. 11, 21, 30, 38, and 45. (Figs. No. 24, 34, 43, 51, and 58.)

Hydrochloric Acid.—The results from these tests are shown in Fig. 7. It will be seen that all of the metals showed considerable corrosion in hydrochloric acid and that chromium was especially

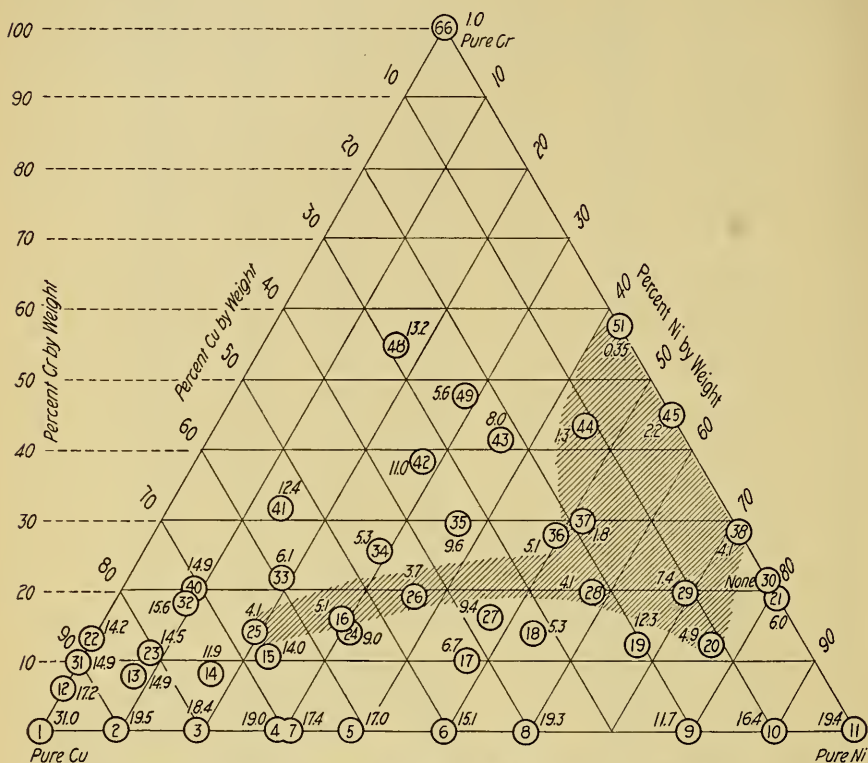


FIG. 8. DIAGRAM SHOWING COMPOSITION OF ALLOYS AND CORROSION IN NORMAL SULFURIC ACID SOLUTION.

soluble in this acid. However, there is a fairly well-defined area in the triangle over which the corrosion was small.

Sulfuric Acid.—The data obtained from the corrosion tests in normal sulfuric acid are shown in Fig. 8. The results do not require any special explanation.

Sodium Hydroxide.—Fig. 9 shows the results obtained by the corrosion tests in normal sodium hydroxide. Copper and nickel show appreciable losses, while chromium is little attacked. In general the losses for the alloys are small.

Ammonium Hydroxide.—The alloys which best resist corrosion by ammonia are near the nickel corner of the triangle and on the chromium-nickel side. The results are shown in Fig. 10.

The copper-chromium and the copper-rich alloys show a selective corrosion in ammonium hydroxide; that is, the copper is removed

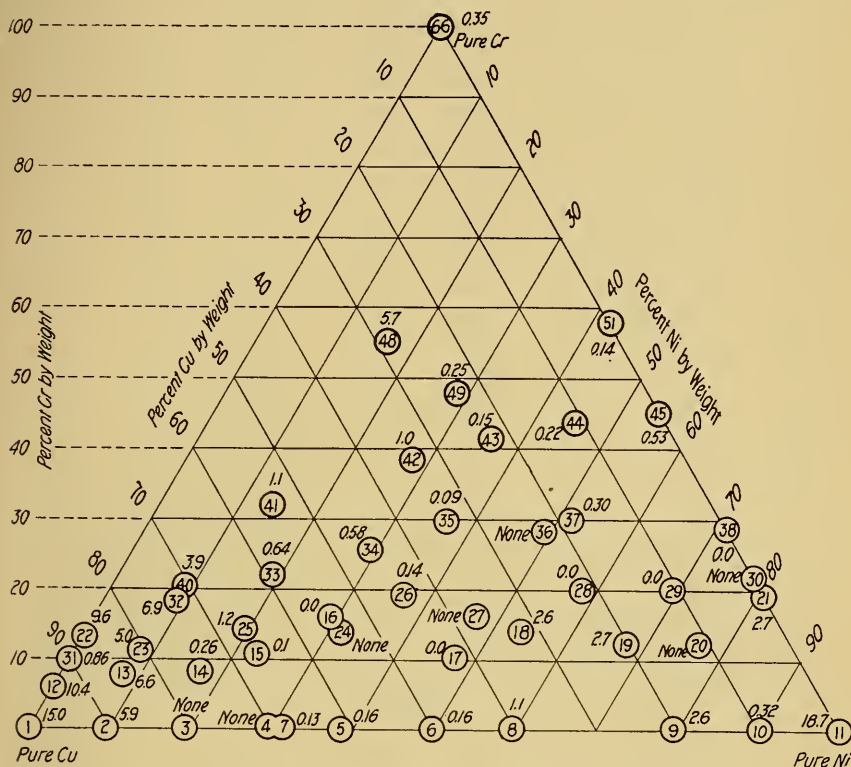


FIG. 9. DIAGRAM SHOWING COMPOSITION OF ALLOYS AND CORROSION IN NORMAL SODIUM HYDROXIDE SOLUTION.

and some of the chromium is left. This fact suggests that good results might be obtained by using ammonia as an etching reagent in the preparation of specimens for microscopic examination.

Sodium Chloride.—Losses in normal sodium chloride solution are, in general, small. These losses are largest for alloys near the copper corner of the triangle. (See Fig. 11.)

Fatty Acids.—Only twenty-four specimens were corroded in the fatty acids. Twelve of the tests were made at 105 degrees C. by heating in an electric oven. The other tests were made at approximately

85 degrees C. by heating on a steam bath. The results are shown in Fig. 12. The greatest losses are near the copper corner of the triangle. This suggests that instead of using a copper container for fatty acids it might be advisable to use an alloy of copper and nickel, or possibly one containing copper and nickel with a little chromium.

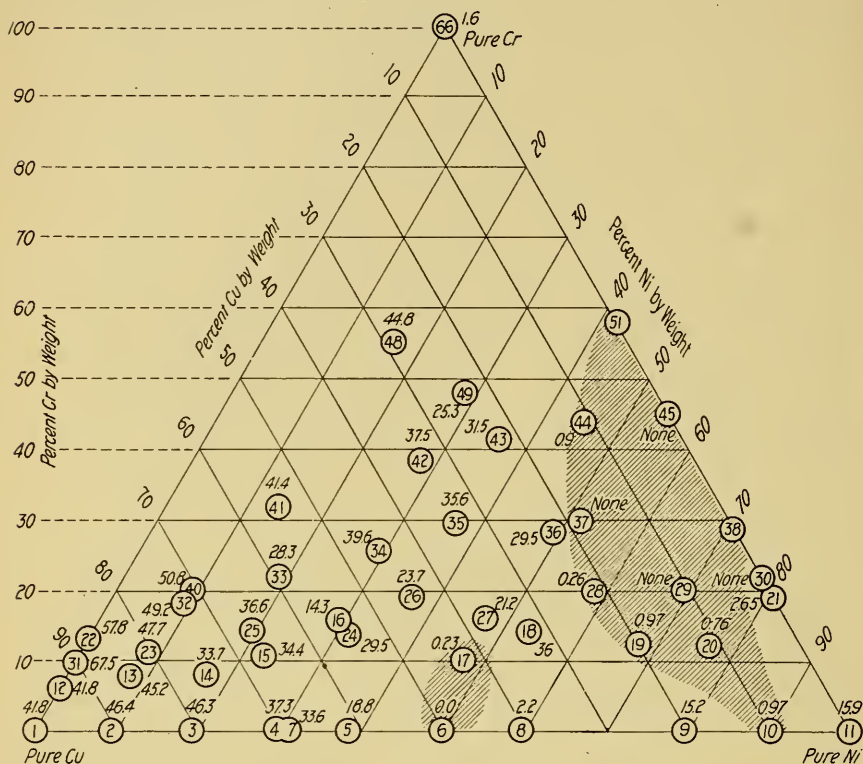


FIG. 10. DIAGRAM SHOWING COMPOSITION OF ALLOYS AND CORROSION IN NORMAL AMMONIUM HYDROXIDE SOLUTION.

Comparison of Corrosions.—Table 6, page 31, shows the relative corrosions in the different reagents which have been used. The values are losses in weight in milligrams per square inch per week.

21. *Conclusions.*—The corrosion tests show that the amount of loss in the different reagents is not proportional to the strengths of the different acids or bases.

So far, it has not been possible to show any definite relation between the relative electromotive forces and corrosion losses.

TABLE 6

COMPARISON OF THE CORROSIONS IN THE DIFFERENT SOLUTIONS

THE VALUES ARE LOSS IN WEIGHT IN MILLIGRAMS PER WEEK PER SQUARE INCH

Number	NaCl	HCl	H ₂ SO ₄	HNO ₃	NaOH	NH ₄ OH	Fatty acids
1	5.08	45.6	31.00	30.80	15.00	41.80	8.90
2	4.18	20.0	19.50	41.40	5.90	46.40	...
3	1.60	34.2	18.40	27.70	None	46.30	5.80
4	1.65	29.4	19.00	25.10	None	37.30	...
5	2.35	26.7	17.00	21.40	0.16	18.80	2.90
6	2.37	21.2	15.10	11.40	0.16	0.00	...
7	0.78	7.1	19.30	22.40	1.10	2.20	...
8	1.71	23.0	17.40	11.40	0.13	33.80	4.00
9	1.71	16.5	11.70	14.80	2.60	15.20	...
10	1.13	18.2	16.40	11.60	0.32	0.97	...
11	None	207.0	19.40	2270.00	18.70	15.90	...
12	4.33	42.8	17.20	12.60	10.40	41.80	10.50
13	2.55	28.6	14.90	10.40	6.60	45.20	...
14	1.89	6.3	11.90	10.80	0.26	33.70	7.70
15	1.53	4.8	14.00	9.90	0.10	34.40	...
16	0.66	6.0	5.10	8.50	0.00	14.30	...
17	0.53	4.6	6.70	10.80	0.00	0.23	None
18	0.71	8.2	5.30	15.10	2.60	3.60	...
19	0.63	7.3	12.30	17.20	2.70	0.97	1.14
20	0.60	10.7	4.90	17.80	None	0.76	...
21	None	19.4	6.00	980.00	2.70	2.65	...
22	4.00	38.3	14.20	13.10	9.60	57.80	...
23	3.65	29.5	14.50	16.30	5.00	47.70	6.40
24	0.76	33.3	9.00	13.40	None	29.50	...
25	2.13	8.2	4.10	10.50	1.20	36.60	6.60
26	0.94	6.2	3.70	7.60	0.14	23.70	3.80
27	0.85	18.8	9.40	6.20	None	21.20	0.87
28	0.37	4.4	4.10	17.70	0.00	0.26	...
29	0.51	6.1	7.40	18.10	0.00	0.00	0.25
30	Too many blowholes.						
31	4.60	30.3	16.40	60.30	0.86	67.50	10.70
32	3.30	10.8	15.60	10.80	6.90	49.20	...
33	2.62	5.0	6.10	7.50	0.64	28.30	7.50
34	1.37	6.6	5.30	4.80	0.58	39.60	...
35	1.05	12.3	9.60	4.80	0.09	35.60	3.80
36	0.37	11.6	5.10	4.90	None	29.50	...
37	...	5.9	1.80	4.90	0.30	None	...
38	0.37	8.1	4.10	492.00	0.00	None	8.30
39	Not prepared.						
40	2.95	15.2	14.90	9.20	3.90	50.80	11.30
41	1.84	7.5	12.40	8.30	1.10	41.40	...
42	1.67	11.5	11.00	7.30	1.00	37.50	2.90
43	1.25	18.6	8.00	4.40	0.15	31.50	...
44	0.45	10.0	1.30	1.02	0.22	0.90	0.34
45	0.40	21.2	2.20	0.60	0.53	0.00	...
46	Not prepared.						
47	Not prepared.						
48	2.45	16.2	13.20	9.70	5.70	44.80	3.25
49	0.87	9.5	5.60	6.40	0.25	25.30	...
50	0.37	19.5	6.40	7.40	1.00	0.90	2.40
51	0.27	31.2	0.35	0.00	0.14	0.00	...
55	0.33	11.9	3.70	7.70	1.00	0.20	...
56	0.27	32.2	1.30	0.35	0.44	0.50	...
66	2.00	16976.4	1.00	0.35	0.35	1.60	1.23

In all cases there are certain fairly well-defined ranges of composition as shown in the diagrams in which the alloys are highly resistant to corrosion.

Generally the ternary alloys are less corroded than the binary, though there are some exceptions.

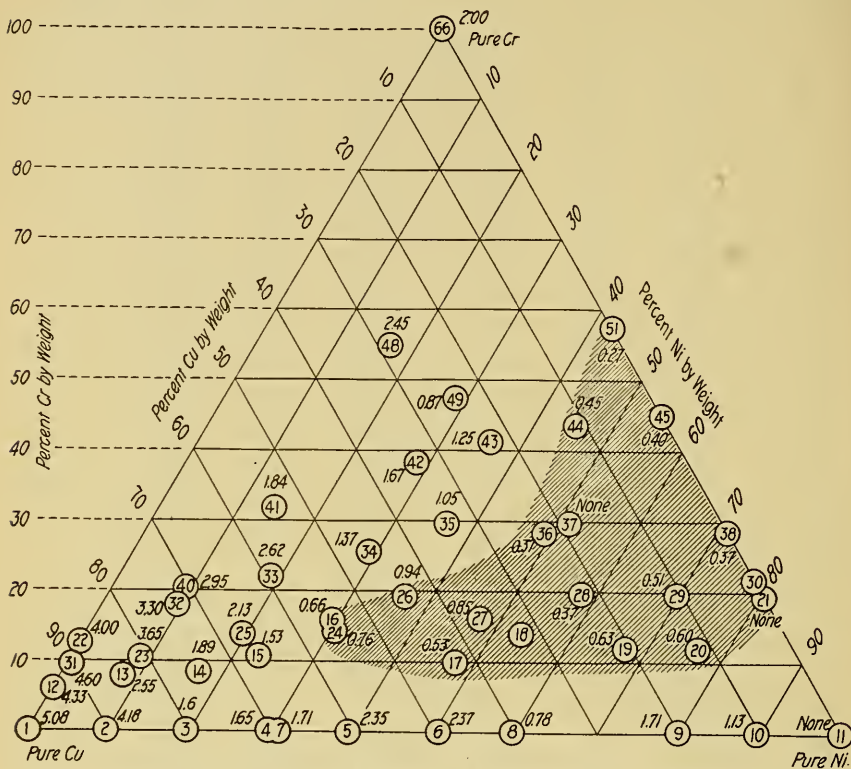


FIG. 11. DIAGRAM SHOWING COMPOSITION OF ALLOYS AND CORROSION IN NORMAL SODIUM CHLORIDE SOLUTION.

V. RELATION BETWEEN CORROSION AND RELATIVE ELECTROMOTIVE FORCES

22. *Purpose of Measurements.*—A series of measurements was made of the relative electromotive forces developed by the different alloys when placed in contact with an electrolyte, to ascertain whether or not any simple relation could be found between the values obtained and the amount of corrosion sustained by the same alloys in the same electrolyte. The measurements were made in a sodium chloride solution and the data obtained is of considerable interest, but it is not sufficient to afford a basis for any general conclusions.

23. *Method and Results of Measurements.*—After some experimentation with other methods the Poggendorf Compensation Method* of measurement was chosen. Briefly described, it consisted in balancing the current from two No. 6 French Auto Special Dry Batteries against the current from a calomel electrode and the alloy in contact

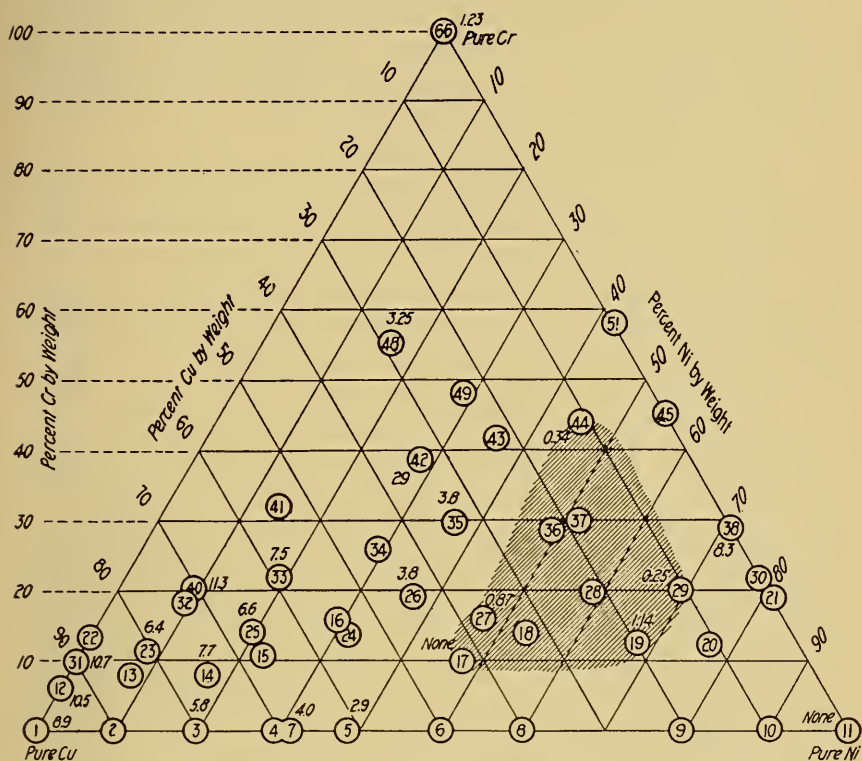


FIG. 12. DIAGRAM SHOWING COMPOSITION OF ALLOYS AND CORROSION IN FATTY ACIDS

with 4 Normal salt solutions. The resistance in the first circuit was 110 ohms. This resistance was varied in the second circuit until the galvanometer showed a zero deflection. A voltmeter, reading 0 to 1.5 volts, was so connected that it showed directly the potential in the calomel electrode-alloy combination. In the different series of measurements all conditions remained the same, except that different alloys were placed in salt solution. Readings were taken at one minute intervals for five minutes. The results are shown in Table 7.

*Stahler, A. Handb. d. Arbeitsmethoden in der anorg. Chem., 1914.

These values are not absolute, but relative, for they were made under the same conditions. The following factors seemed to influence the results: (a) The condition of the specimen as regards polishing. (b) The depth to which the piece was immersed in the solution. The latter factor was active until the piece was at least one-half covered; after that the value did not seem to change even upon complete immersion. (c) The time that the solution and specimen were left in contact, especially if any current was allowed to flow through the system. It is believed that this effect was caused by the deposition of gases upon the specimen or possibly by gases already dissolved in them. (d) Any moving of the specimen or solution.

It was noticed that the alloys which contained high percentages of copper gave values which increased with time, whereas those containing high percentages of nickel gave values which decreased with time. These results together with the composition of the alloys are shown diagrammatically in Fig. 13. Those alloys which showed an increase have been marked with a plus sign and those which showed a decrease have been marked with a minus sign. Alloys Nos. 17, 18, and 19 seem irregular, but they contained blowholes; and if there was any concentration of the low melting constituent (copper) at the blowholes, it might be expected that they would behave like the copper-rich alloys. No. 24 is the only one which did not show a change in the five minute interval, although the changes shown by some of the others were very small. From an examination of the corrosion tables it may be seen that the alloys which showed little or no change in relative electromotive force were not immune to corrosion in the different solutions. Therefore, it does not seem that resistance to corrosion can be predicted from the fact that the relative electromotive force of the alloy in contact with an electrolyte either remained constant or showed little variation.

In the case of the corrosions in normal salt solutions it was noticed that, almost without exception, the alloys which had shown an increase in the relative electromotive force were the ones which showed a turbidity of the corroding solution, while those which had given decreasing values remained clear. Similarly, in the corrosions in ammonium hydroxide the solutions were colored a deep blue in the case of the alloys which had shown increases in the relative electromotive forces, while the others remained practically colorless. However, when the corrosion specimens were weighed, it was found that there had been losses in the salt solutions, which had remained clear,

and in the ammonium hydroxide solutions, which had remained colorless.

From the results obtained it is not felt that any safe conclusions can be drawn as to the possibility of predicting the corrodibility of an alloy from such a series of measurements of the relative electromotive forces or from the changes in these values with time.

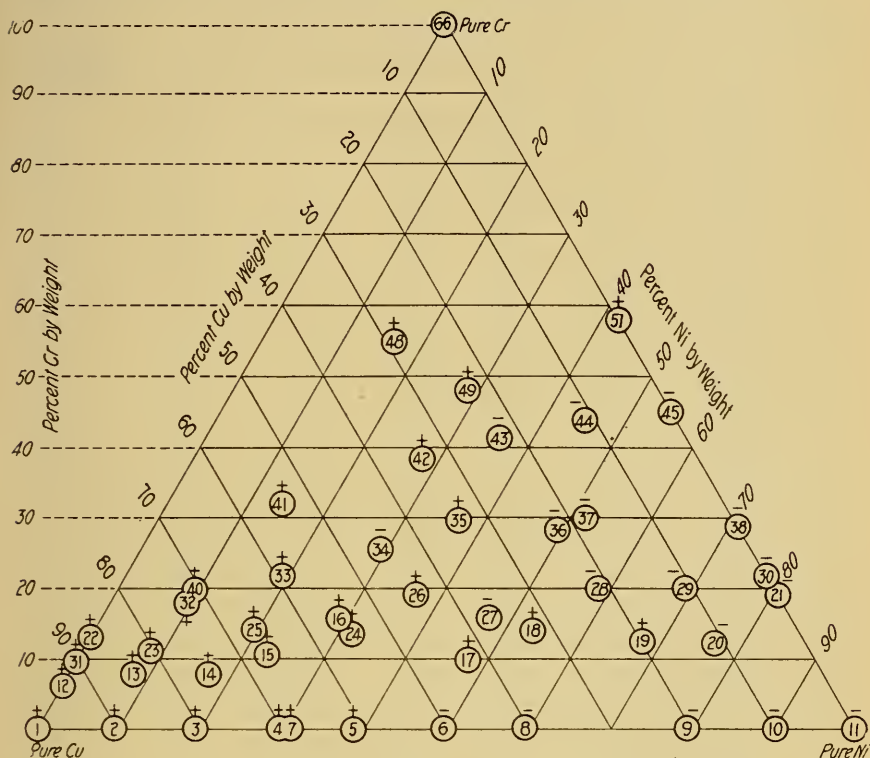


FIG. 13. DIAGRAM SHOWING COMPOSITION OF ALLOYS AND THEIR CHANGES IN ELECTROMOTIVE FORCE IN A 4 NORMAL SODIUM CHLORIDE SOLUTION.

VI. THERMAL ANALYSIS AND MICROSCOPIC EXAMINATION.

24. *Difficulties of Thermal Analysis.*—The very high melting points of the metals and alloys of this system, their great susceptibility to oxidation and absorption of impurities at high temperatures, and the extreme difficulty of finding suitable materials for pyrometers, tubes, and crucibles, all combined to make the study of freezing point curves extremely difficult, and, so far as present developments are concerned, almost impossible.

TABLE 7

RELATIVE ELECTROMOTIVE FORCES OF THE DIFFERENT ALLOYS IN CONTACT WITH
4 NORMAL SALT SOLUTIONS

Number	Relative Electromotive Force After						Remarks
	1 min.	2 min.	3 min.	4 min.	5 min.	Change	
1	...	...	...	...	0.364	...	Increasing
2	0.328	0.328	0.334	0.331	0.331	+ .003	
3	0.328	0.330	0.331	0.333	0.334	+ .006	
4	0.311	0.316	0.317	0.318	0.319	+ .009	
5	0.314	0.318	0.315	0.318	0.315	+ .001	
6	0.309	0.302	0.296	0.295	0.293	— .016	
7	0.313	0.315	0.315	0.318	0.318	+ .005	
8	0.305	0.300	0.300	0.297	0.296	— .009	
9	0.295	0.295	0.290	0.280	0.280	— .015	
10	0.318	0.309	0.300	0.294	0.289	— .029	Pure nickel
11	0.294	0.294	0.290	0.287	0.287	— .007	
12	0.338	0.341	0.345	0.346	0.348	+ .010	
13	0.345	0.343	0.348	0.348	0.350	+ .005	
14	0.340	0.341	0.341	0.339	0.339	— .001	
15	0.348	0.345	0.344	0.342	0.341	— .007	
16	0.342	0.336	0.333	0.332	0.330	— .012	
17	0.310	0.309	0.306	0.306	0.304	— .006	
18	0.307	0.304	0.303	0.303	0.303	— .004	
19	0.306	0.312	0.315	0.316	0.318	+ .012	
20	0.331	0.325	0.315	0.309	0.300	— .031	
21	0.294	0.282	0.279	0.276	0.275	— .019	
22	0.357	0.360	0.363	0.363	0.363	+ .006	
23	0.345	0.347	0.348	0.349	0.349	+ .004	
24	0.307	0.306	0.307	0.307	0.307	+ .000	
25	0.315	0.319	0.322	0.324	0.325	+ .010	
26	0.336	0.339	0.339	0.339	0.339	+ .003	
27	0.330	0.326	0.320	0.320	0.320	— .010	
28	0.355	0.344	0.339	0.332	0.330	— .025	
29	0.360	0.349	0.341	0.338	0.332	— .028	
30	0.325	0.320	0.329	0.331	0.324	— .001	Not prepared
31	0.339	0.340	0.345	0.346	0.349	+ .010	
32	0.330	0.337	0.339	0.340	0.341	+ .011	
33	0.341	0.347	0.346	0.346	0.346	+ .005	
34	0.340	0.339	0.339	0.338	0.338	— .002	
35	0.333	0.333	0.334	0.334	0.334	+ .001	
36	0.334	0.330	0.329	0.320	0.320	— .014	
37	0.332	0.330	0.320	0.319	0.315	— .017	
38	0.316	0.310	0.305	0.300	0.296	— .020	
39	...	...	...	...	...	...	
40	0.342	0.346	0.352	0.355	0.355	+ .013	Not prepared
41	0.350	0.351	0.352	0.355	0.355	+ .005	
42	0.331	0.335	0.339	0.339	0.339	+ .008	
43	0.366	0.365	...	0.350	0.346	— .020	
44	0.314	0.300	0.290	0.286	0.286	— .028	
45	0.322	0.315	0.309	0.302	0.300	— .022	
46	...	...	...	...	...	...	
47	...	...	...	...	...	...	
48	0.338	0.345	0.349	0.351	0.350	+ .012	
49	0.345	0.347	0.349	0.349	0.349	+ .004	
50	0.298	0.299	0.298	0.295	0.295	— .003	Not prepared
51	0.468	0.473	0.476	0.480	0.480	+ .012	

Note: The other specimens have either not been prepared or have not been tested.

The handicaps were overcome in part by the construction of a specially designed granular carbon resistance furnace which gave sufficiently high temperatures for the melting of the alloys. Fairly satisfactory crucibles were made of fused magnesia, which material served also as a satisfactory refractory for parts of the furnace. By means of these crucibles, melts of about 200 grams could be made.

The greatest difficulty, however, was encountered in attempting to find a protecting tube for the thermo couple which would withstand the severe conditions to which it would be exposed. Quartz tubes are good for low temperatures, but they cannot be used at the melting point of chromium. Porcelain tubes, such as were used by Hindrichs, stand slightly higher temperatures than quartz without softening, but they break easily and are attacked by chromium. An attempt was made to use an alundum tube, but it broke in the first melt. Since it was known that the fused magnesia crucibles were little attacked by chromium, it was thought that an insulating tube of the same material might be satisfactory. Some tubes were moulded, dried, and heated to about 1500 degrees C. They became hard and dense, but bent during the heating. It appears that in heating the magnesia to 1500 degrees C. it passes through a semi-fused state at which time sintering takes place, but at that same time the tubes bend so badly that they can not be used. Some magnesia tubes were moulded in which a solution of magnesium chloride was used as the binding material. These tubes, after drying at 105 degrees C., were hard and looked promising, but when heated they became brittle and crumbled to pieces at 800 to 900 degrees C. So far, all attempts to prepare satisfactory magnesia insulating tubes have failed. Because of so many difficulties and a limited amount of time, it was thought best to omit this part of the work for the time being, reserving it for later investigation.

25. *Heat Treatment.*—As was explained in the introduction it has not been considered advisable to attempt much in the way of heat treatment of the alloys until the equilibrium diagrams have been more thoroughly established.

Some annealing tests have been carried out and are mentioned at the close of this chapter.

26. *Microscopic Examination.*

a. *General Discussion.*—The metallurgical microscope used in the examination of these alloys was a Leitz "Micrometallograph." For the most part apochromatic objectives were used with projection eyepieces. Many difficulties were met with in the preparation of the

different alloys for microscopic study. No two alloys had exactly the same composition. It is almost equally true that no two of the alloys would give the best results by the same methods of polishing, etching, staining, and photographing. Thus each specimen became a research problem in itself.

b. *Etching Reagents*.—A study of the different etching reagents, stains, etc., described in the literature, was made in an endeavor to find means of identifying the different constituents in the alloys. Descriptions of the two reagents which have been found most effective follow:

Ferric Chloride in Hydrochloric Acid.—This solution consisted of 1 per cent FeCl_3 dissolved in 1:1 HCl . It has been found very useful. In some cases it was found necessary to dilute it with one or more volumes of water because the etching was too rapid. An attempt was made to secure colorations by using a mixture of 25 cc. of this solution with 25 cc. of glycerine, 2 grams of resorcin, and 50 cc. of water, but it did not give any better results.

Iodine.—Iodine has been used as the tincture and in a solution of potassium iodide. These solutions can be used to etch the specimen in some cases, but they have more value in staining the specimen after it has been etched with some other reagent. Iodine stains copper or the copper rich constituent, giving it a dark appearance; because it has little effect on either chromium or nickel, it has proved the best method for identifying the different constituents in these alloys.

Other reagents tried, but with little success, were alkaline potassium permanganate, a mixture of picric and nitric acids in amyl and ethyl alcohols, picric acid in alcohol, sodium picrate in alcohol, and tartaric acid in water.

Additional information regarding the etching and staining used may be obtained by a study of the microphotographs. (Figs. No. 14 to 65.)

c. *Results*.—Some of the results obtained from the microscopic examination have been referred to in the previous discussions. In general the data given with the microphotographs are sufficient to explain them. The composition given is in weight per cent as found by analysis, unless otherwise stated.

Chromium-Nickel Alloys.—The microscopic examination of the alloys of chromium and nickel seems to confirm Voss' conclusions:

First, that chromium and nickel form a series of solid solutions (mixed crystals) in the alloys containing from 100 to 50 per cent

of nickel. (See Figs. 24, 34, 43, 51, and 58. Alloys No. 11, 21, 30, 38, and 45.)

Second, that chromium and nickel form a eutectic which contains about 42 per cent of nickel. Guertler does not think that the metals form a true eutectic, but calls it a pseudoeutectic. Fig. 63, Alloy No. 51, is of an alloy having approximately 42 per cent of nickel and which appears to have a eutectic-like structure. Specimen No. 56, Fig. 65, which should contain more chromium, shows what appears to be the eutectic structure and an excess of chromium. Although alloys containing higher percentages of chromium have been prepared, they have not been analyzed and microphotographs of them have not been made. The reason for this has been the difficulty of cutting proper samples from the extremely hard alloys.

Copper-Nickel Alloys.—The microphotographs of the alloys of copper and nickel agree very well with those obtained by Guertler and Tammann, especially for those obtained under similar conditions; namely, with slow cooling. The alloys containing more than 80 per cent of nickel show large polyhedral crystals. Microphotographs of the pure metals and their alloys are shown in Figs. 14 to 24.

Solubility of Chromium in Copper.—The tendency for chromium or a chromium-rich constituent to separate in alloys of chromium and copper has already been discussed. This effect may be seen in Alloys Nos. 12, 22, and 31; Figs. 25, 35, and 44, and in Figs. 2 and 3. The chromium or chromium-rich constituent shows either in relief or as the light part of the photograph, if the specimen was stained with iodine.

Effect of Nickel on the Solubility of Chromium.—The effect of the addition of nickel to alloys of chromium, copper, and nickel may be studied in the microphotographs of Alloys Nos. 22 to 30, inclusive, Figs. 35 to 43, in which series there is an increase in nickel and a decrease in copper. The separation of the chromium or of the chromium-rich constituent is apparent in the Alloys Nos. 22 to 26. Those containing larger percentages of nickel are more homogeneous. In No. 29, Fig. 42, well-defined polyhedral crystals characteristic of a solid solution are shown. Similar effects will be seen if other series, such as Nos. 12 to 21, Figs. 25 to 34, and Nos. 31 to 38, Figs. 44 to 51 are studied. The alloys become practically homogeneous when the amount of nickel is more than three times the amount of copper present.

Crystals in Nickel-Rich Alloys.—It must be remembered that the alloys studied were intended to have variations of 10 per cent in their different constituents and it is not possible to say at exactly what nickel content the binary alloys begin to show well-defined polyhedral crystals. From the specimens examined, it is evident that in the case of the binary alloys those which contain as much as 80 per cent of nickel show such crystals. On the other hand, the ternary alloys show them if they contain as much as 70 per cent of nickel. Of course, the structures which have been obtained in these alloys represent what may be expected if the castings are slowly cooled, but they do not necessarily show what structure would be produced by quenching.

Annealing Tests.—Small pieces of Alloys Nos. 22 to 38, inclusive, were packed in amorphous silica in an iron pipe, $1\frac{3}{4}$ by 5 inches in dimensions and closed at both ends by caps. The pipe and contents were placed in an electric furnace and heated at a temperature of approximately 900 degrees C. for at least twenty-four hours. The specimens were repolished and examined microscopically. From a brief examination it seems that there was not any very noticeable change in the structure.



FIG. 14. ALLOY No. 1. x 40.
CU. 100%
ETCHED IN 1% FeCl_3 IN 1:1 HCl.

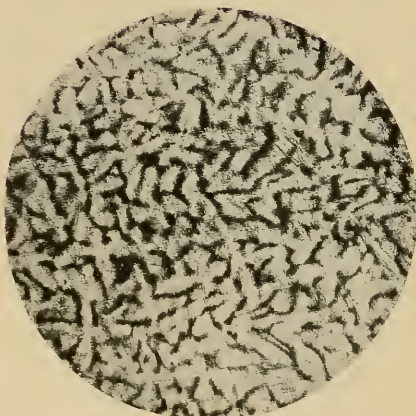


FIG. 15. ALLOY No. 2. x 40.
90.84% CU, 9.06% NI BY WEIGHT.
ETCHED IN 1% FeCl_3 IN 1:1 HCl.

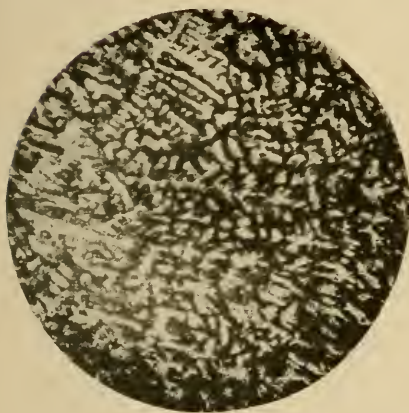


FIG. 16. ALLOY No. 3. x 40.
81.07% CU, 18.76% NI, BY WEIGHT.
ETCHED IN 1% FeCl_3 IN 1:1 HCl.



FIG. 17. ALLOY No. 4. x 40.
71.16% CU, 28.46% NI BY WEIGHT.
ETCHED IN 1% FeCl_3 IN 1:1 HCl.

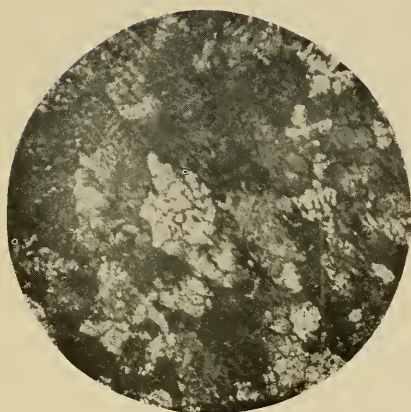


FIG. 18. ALLOY No. 5. x 40.
61.63% CU, 38.25% NI BY WEIGHT.
ETCHED IN 1% FeCl_3 IN 1:1 HCL.



FIG. 19. ALLOY No. 6. x 40.
48.96% CU, 49.90% NI BY WEIGHT.
ETCHED IN 1% FeCl_3 IN 1:1 HCL.

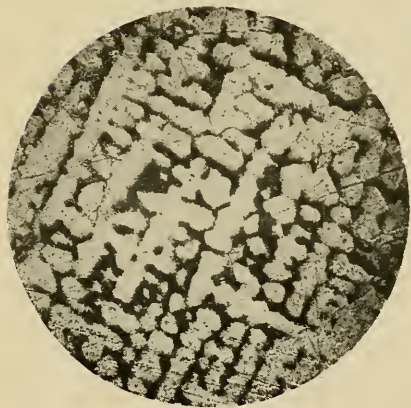


FIG. 20. ALLOY No. 7. x 40.
69.13% CU, 30.59% NI.
ETCHED IN 1% FeCl_3 IN 1:1 HCL.

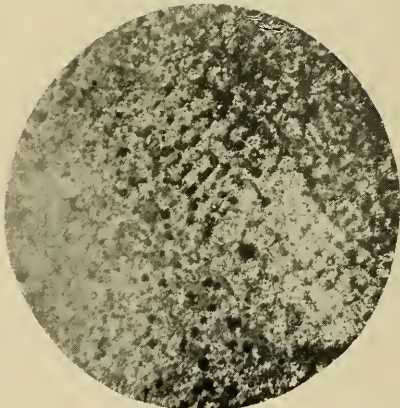


FIG. 21. ALLOY No. 8. x 40.
41.14% CU, 58.27% NI.
ETCHED IN 1% FeCl_3 IN 1:1 HCL.

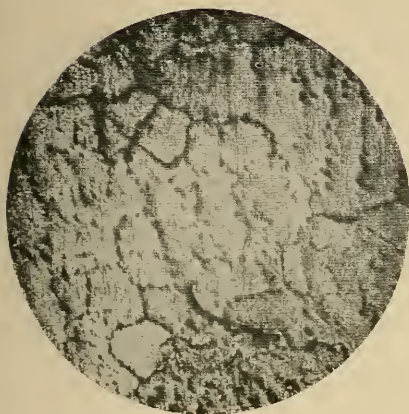


FIG. 22. ALLOY No. 9. x 40.
20.65% Cu, 79.35% Ni.
ETCHED IN 1% FeCl_3 IN 1:1 HCl.

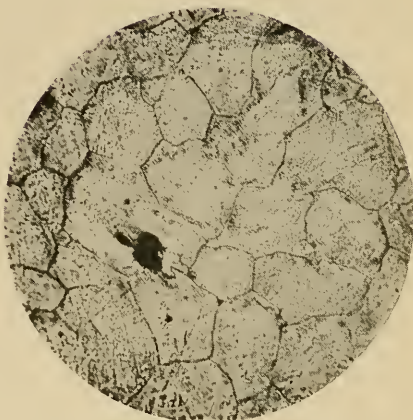


FIG. 23. ALLOY No. 10. x 40.
10.57% Cu, 88.90% Ni.
ETCHED IN 1% FeCl_3 IN 1:1 HCl.

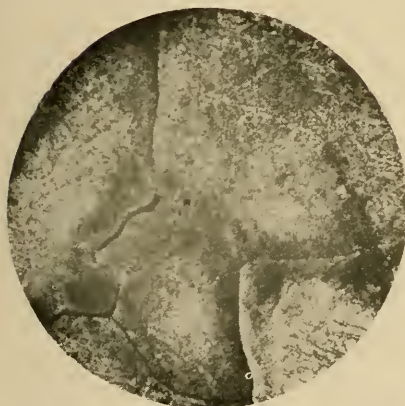


FIG. 24. ALLOY No. 11. x 40.
99.66% Ni.
ETCHED IN 1% FeCl_3 IN 1:1 HCl.

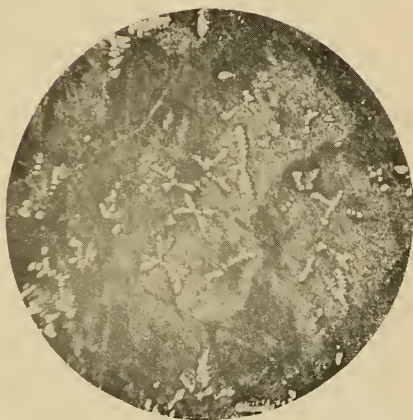


FIG. 25. ALLOY No. 12. x 40.
6.08% Cr, 94.20% Cu.
ETCHED IN 1% FeCl_3 IN 1:1 HCl.

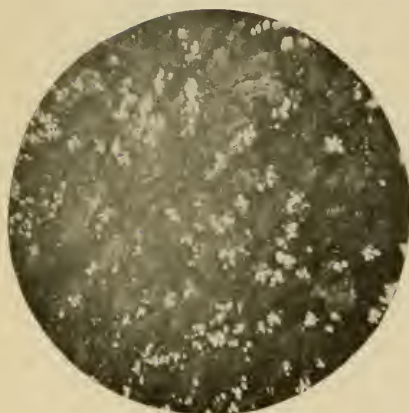


FIG. 26. ALLOY No. 13. x 40.
7.8% CR, 84.4% CU, 9.3% NI
ETCHED IN 1% FeCl_3 IN 1:1 HCl.



FIG. 27. ALLOY No. 14. x 40.
8.25% CR, 74.63% CU, 16.05% NI.
ETCHED IN 1% FeCl_3 IN 1:1 HCl.

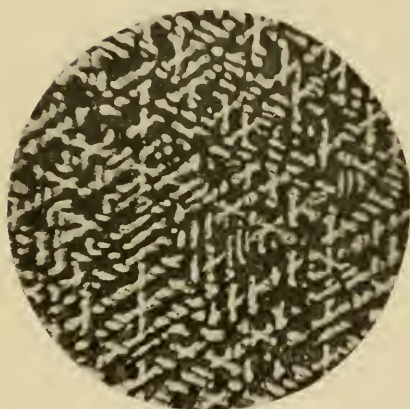


FIG. 28. ALLCY No. 15. x 40.
10.6% CR, 66.3% CU, 22.9% NI.
ETCHED IN 1% FeCl_3 IN 1:1 HCl.



FIG. 29. ALLOY No. 16. x 40.
15.9% CR, 54.7% CU, 29.4% NI.
ETCHED IN 1% FeCl_3 IN 1:1 HCl.



FIG. 30. ALLOY No. 17. x 40.
10.1% CR, 42.6% CU, 48.0% NI.
ETCHED IN 1% FeCl_3 IN 1:1 HCL.

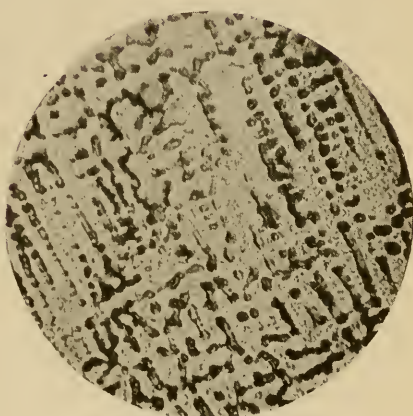


FIG. 31. ALLOY No. 18. x 40.
13.97% CR, 32.69% CU, 54.16% NI.
ETCHED IN 1% FeCl_3 IN 1:1 HCL.

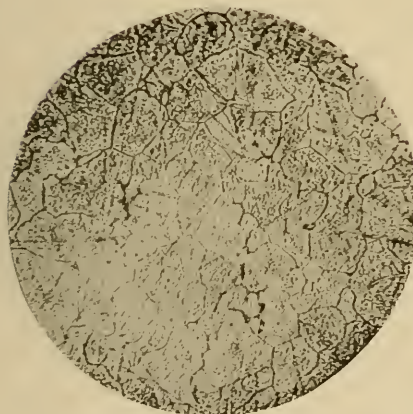


FIG. 32. ALLOY No. 19. x 40.
11.80% CR, 20.85% CU, 66.25% NI.
ETCHED IN 1% FeCl_3 IN 1:1 HCL.

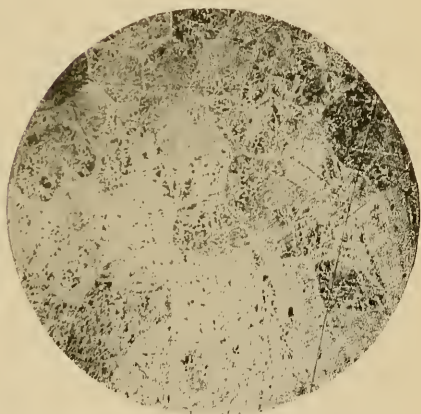


FIG. 33. ALLOY No. 20. x 40.
11.90% CR, 11.83% CU, 76.27% NI.
ETCHED IN 1% FeCl_3 IN 1:1 HCL.

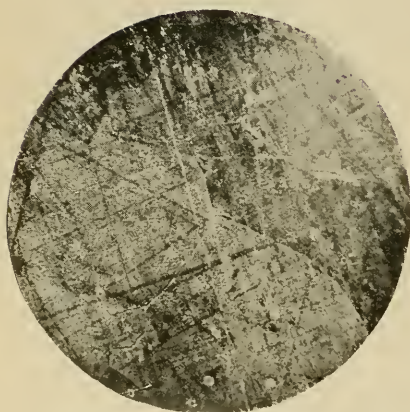


FIG. 34. ALLOY No. 21. x 40.
19.37% Cr, 78.99% Ni.
ETCHED IN 1% FeCl_3 IN 1:1 HCl.



FIG. 35. ALLOY No. 22. x 40.
13.15% Cr, 87.93% Cu.
NOT ETCHED. STAINED IN I IN KI.

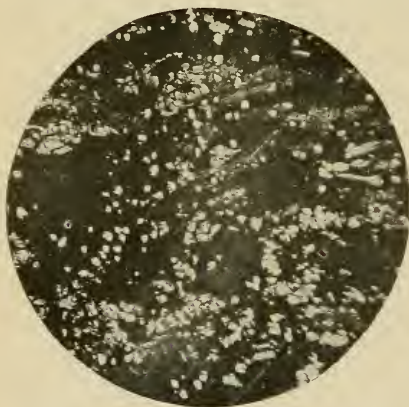


FIG. 36. ALLOY No. 23. x 40.
10.56% Cr, 80.58% Cu, 9.38% Ni.
ETCHED LIGHTLY IN 1% FeCl_3 IN 1:1
HCl.



FIG. 37. ALLOY No. 24. x 40.
14.56% Cr, 56.30% Cu, 29.24% Ni.
ETCHED AND THEN STAINED IN I IN KI.



FIG. 38. ALLOY No. 25. x 40.
13.62% CR, 66.92% CU, 19.20% NI.
ETCHED AND THEN STAINED IN I IN KI.



FIG. 39. ALLOYED No. 26. x 40.
19.30% CR, 44.08% CU, 36.34% NI.
NOT ETCHED, BUT STAINED IN I IN KI.



FIG. 40. ALLOY No. 27. x 40.
15.99% CR, 36.70% CU, 47.31% NI.
STAINED IN I IN ALCOHOL.

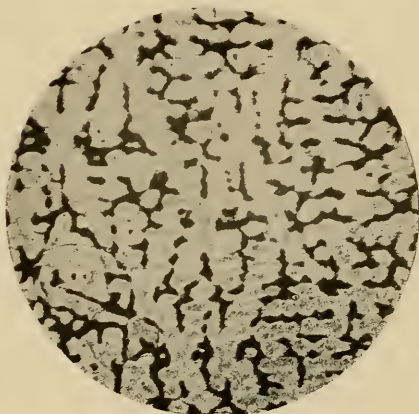


FIG. 41. ALLOY No. 28. x 40.
19.86% CR, 22.20% CU, 57.36% NI,
STAINED IN I IN ALCOHOL.



FIG. 42. ALLOY No. 29. x 40.
19.64% CR, 10.88% CU, 68.62% NI.
ETCHED IN 1% FeCl_3 IN 1:1 HCL.

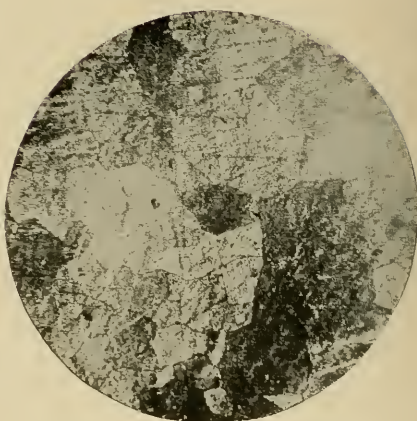


FIG. 43. ALLOY No. 30. x 40.
21.52% CR, 76.95% NI.
ETCHED IN 1% FeCl_3 IN 1:1 HCL.

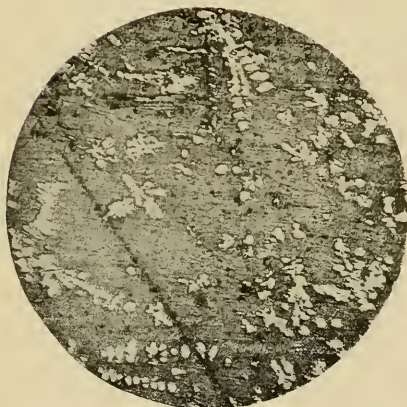


FIG. 44. ALLOY No. 31. x 40.
9.89% CR, 89.82% CU.
NOT ETCHED. RELIEF POLISHING.



FIG. 45. ALLOY No. 32. x 40.
17.66% CR, 73.63% CU, 8.55% NI.
NOT ETCHED. RELIEF POLISHING.

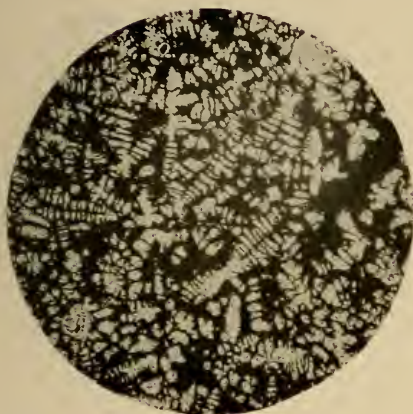


FIG. 46. ALLOY No. 33. x 40.
22.00% CR, 59.62% CU, 19.48% NI.
ETCHED IN I IN KI.



FIG. 47. ALLOY No. 34. x 40.
25.10% CR, 45.70% CU, 29.52% NI.
ETCHED, THEN STAINED IN I IN KI.



FIG. 48. ALLOY No. 35. x 40.
29.46% CR, 33.76% CU, 36.78% NI.
ETCHED, THEN STAINED IN I IN KI.



FIG. 49. ALLOY No. 36. x 40.
28.10% CR, 22.58% CU, 48.42% NI.
ETCHED, THEN STAINED IN I IN KI.

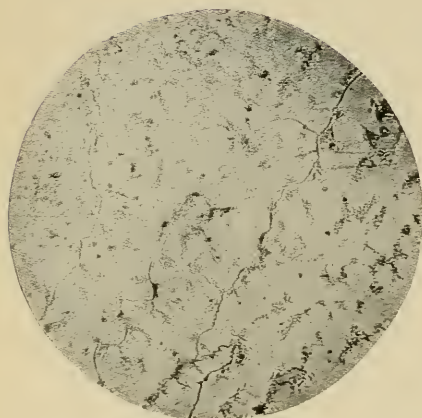


FIG. 50. ALLOY No. 37. x 40.
29.70% CR, 10.90% CU, 58.12% NI.
ETCHED IN 1% FeCl_3 IN 1:1 HCL.



FIG. 51. ALLOY No. 38. x 40.
28.44% CR, 71.56% NI.
ETCHED IN 1% FeCl_3 IN 1:1 HCL.

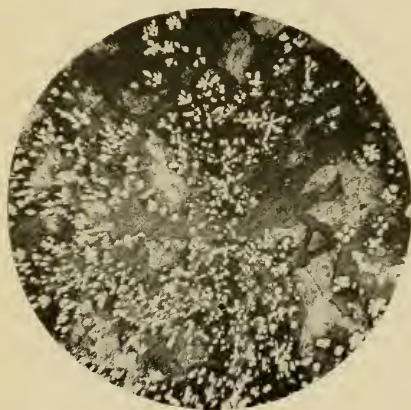


FIG. 52. ALLOY No. 39. x 40.
APPROX. 35.29% CR, 64.71% CU.



FIG. 53. ALLOY No. 40. x 40.
19.93%CR, 70.57% CU, 8.99% NI.
NOT ETCHED. RELIEF POLISHING.



FIG. 54. ALLOY No. 41. x 40.
31.63% Cr, 54.16% Cu, 14.21% Ni.
STAINED IN I IN ALCOHOL.

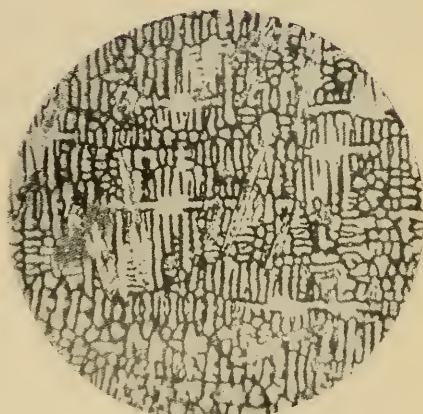


FIG. 55. ALLOY No. 42. x 40.
38.16% Cr, 33.60% Cu, 26.78% Ni.
ETCHED, THEN STAINED IN I IN
ALCOHOL.

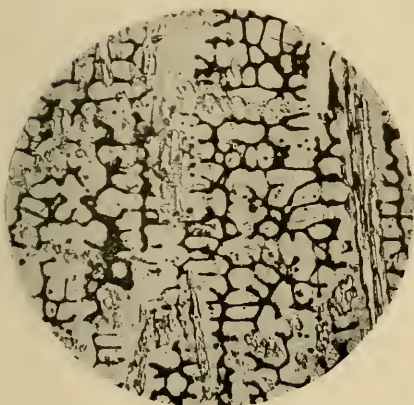


FIG. 56. ALLOY No. 43. x 40.
41.32% Cr, 22.68% Cu, 34.60% Ni.
ETCHED, THEN STAINED IN I IN
ALCOHOL.

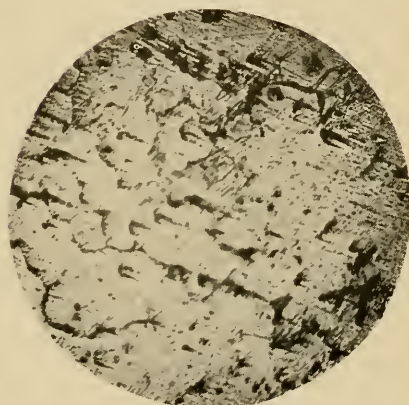


FIG. 57. ALLOY No. 44. x 40.
43.30% Cr, 11.02% Cu, 46.46% Ni.
ETCHED, THEN STAINED IN I IN
ALCOHOL.

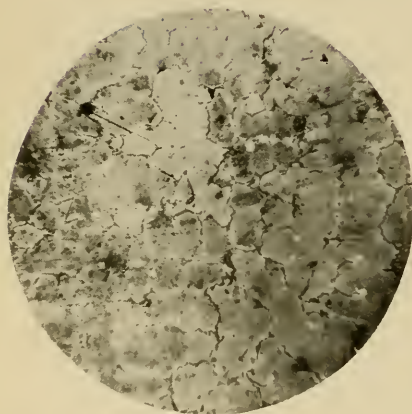


FIG. 58. ALLOY NO. 45. x 40.
44.93% CR, 56.55% NI.
ETCHED IN AQUA REGIA.

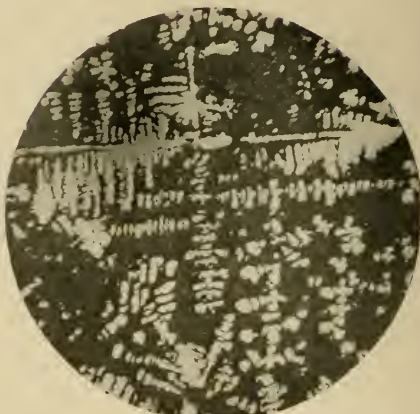


FIG. 59. ALLOY NO. 48. x 60.
54.92% CR, 28.42% CU, 17.12% NI.
STAINED IN I IN ALCOHOL.



FIG. 60. ALLOY NO. 49. x 40.
47.54% CR, 24.12% CU, 26.28% NI.
POLISHED. NOT ETCHED.

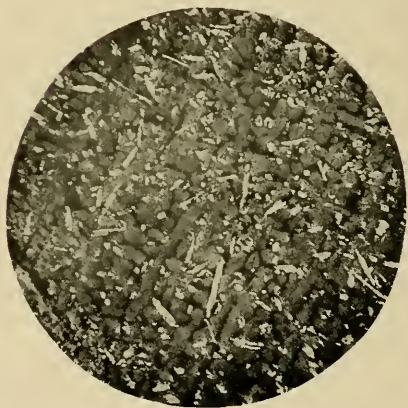


FIG. 61. ALLOY NO. 49. x 40.
47.54% CR, 24.12% CU, 26.28% NI.
STAINED IN I IN KI.

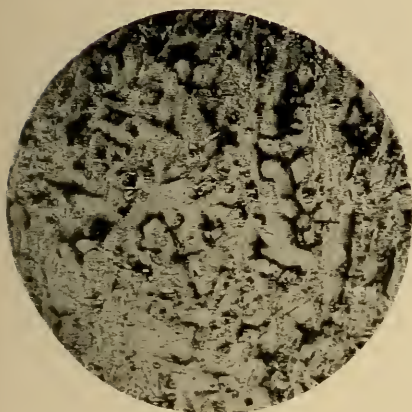


FIG. 62. ALLOY NO. 50. x 40.
APPROX. 46.6% Cr, 11.4% Cu, 42.0% Ni.

STAINED IN I IN ALCOHOL.

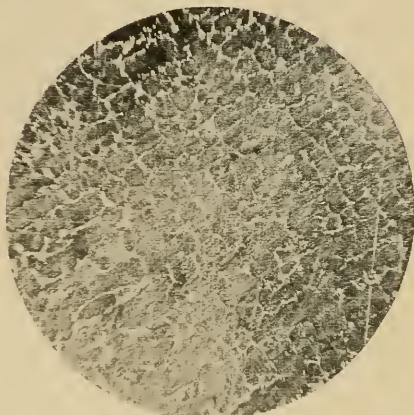


FIG. 63. ALLOY NO. 51. x 40.
57.40% Cr, 41.66% Ni.
ETCHED IN AQUA REGIA.



FIG. 64. ALLOY NO. 55. x 40.
APPROX. 56.6% Cr, 11.5% Cu, 31.9% Ni.

ETCHED IN AQUA REGIA.

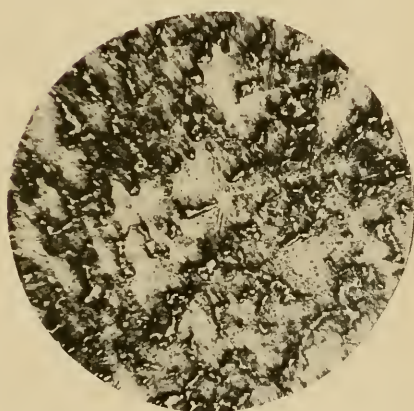


FIG. 65. ALLOY NO. 56. x 40.
APPROX. 57% Cr, 43% Ni.

ETCHED IN 1% FeCl_3 IN 1:1 HCl.
THEN STAINED IN I IN ALCOHOL.

APPENDIX

HISTORICAL REVIEW

In this review it is intended to give only the more important results of previous investigations and references to the original publications.

1. *Copper-Nickel Alloys*.—Christoffe and Bouilhet* prepared alloys containing 50 per cent of copper and 50 per cent of nickel, and 85 per cent of copper and 15 per cent of nickel, and observed some of their properties.

In 1896 H. Gautier† made a more extensive study of these alloys. He determined their freezing points and concluded that they formed a definite chemical compound having the formula CuNi which melted at 1340 degrees C. The original paper was presented by H. Moissan.

Heycock and Neville‡ in their work on the “Complete Freezing-point Curves of Binary Alloys containing Silver or Copper together with Another Metal” tried the effect of the addition of small amounts of nickel upon the freezing point of copper and found that the freezing point was raised from 1080 to 1110 degrees C. by the addition of 4.5 per cent of nickel.

Kurnakoff and Schemtschny§ prepared alloys from electrolytic copper and nickel, determined their freezing points, plotted the freezing-point curve, studied the structure of the different specimens, and pointed out certain similarities to the alloys of iron-copper, cobalt-copper, and copper-nickel. They took 1484 degrees C. as the melting point of nickel which is now known to be too high.

Guertler and Tammann§ in their investigation of the alloys of copper and nickel showed that there was no break in either the liquidus or solidus curves. This showed that these alloys do not form a definite chemical compound as had been claimed by H. Gautier. They made also both magnetic and microscopic examinations and showed the effect of heat on the magnetic properties as well as the effect of the rate of cooling on the grain and crystal size.

*Christoffe and Bouilhet, *Bul. Soc. Chem.*, Vol. 26, p. 419, 1876.
Compt. rend., Vol. 83, p. 29, 1876.

†H. Gautier, *Compt. rend.*, Vol. 123, p. 172, 1896.

‡Heycock and Neville, *Philos. Trans.*, 189A, p. 25, 1897.

§Kurnakoff and Schemtschny, *Z. anorg. Chem.*, Vol. 54, p. 149, 1907.

§Guertler and Tammann, *Z. anorg. Chem.*, Vol. 52, p. 25, 1907.

The following year (1908) Victor E. Tafel* published the results of his studies of the constitution of the binary system copper-nickel. His work seems to be the best that has been published. For convenience his results are shown diagrammatically in Fig. 66. He obtained higher values for both the liquidus and the solidus curves than did Guertler and Tammann, but that may be attributed to the fact that they used nickel which contained a considerable amount of impurities (0.47 per cent Fe, 1.86 per cent Co.) which would lower the freezing points.

E. Vigouroux†, using pure metals especially free from cobalt, prepared a series of copper-nickel alloys, but he was unable to detect any indication of definite chemical compounds by chemical investigation or by a study of the electromotive forces in the cells $\text{Ni} - \frac{\text{N}}{1}$, $\text{NiSO}_4\text{—CuNi}$ alloy and $\text{Cu} - \frac{\text{N}}{1}$, $\text{NiSO}_4\text{—CuNi}$ alloy. His results by this method agree with those obtained by the cooling-curve method.

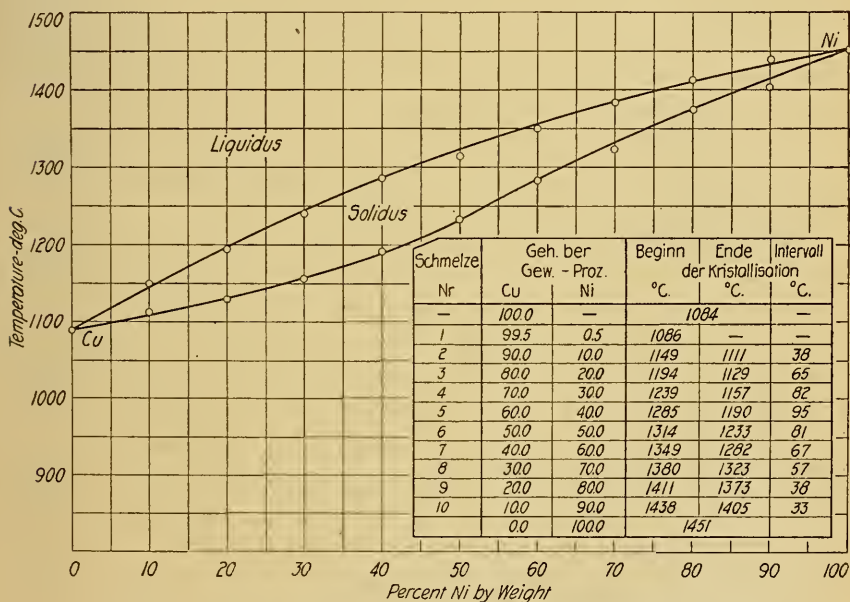


FIG. 66. COPPER-NICKEL EQUILIBRIUM DIAGRAM AFTER TAFEL.

*Victor E. Tafel, *Metallurgie*, Vol. 5, p. 348, 1908.

†E. Vigouroux, *Compt. rend.*, Vol. 159, p. 1378, 1909.

David H. Browne* secured U. S. patent 934,278 (Sept. 14, 1910) for the manufacture of nickel and copper-nickel alloys by electrically fusing compounds of the metals as sulfide matte with lime (CaO), forming calcium sulfide (CaS), sulfur dioxide (SO_2), and an alloy of the metals.

2. *Chromium-Copper Alloys*.—H. Moissan† prepared an alloy of chromium and copper which contained about 0.5 per cent of chromium. It was more resistant to humid air than was copper and took a beautiful polish.

H. Goldschmidt‡ has described an alloy of chromium and copper containing 10 per cent of chromium and having the color of copper, but being harder. The Goldschmidt Thermit Company¶, 90 West Street, New York, now offers for sale an alloy of chromium-copper containing 10 per cent of chromium. The alloy is made by the aluminothermic method. This alloy is discussed more fully in Chapter II.

Hamilton and Smith§ heated chromium oxide and metallic copper in a carbon crucible and in the presence of carbon, by which process they obtained an alloy of gray-red color and of a hardness which placed it next to the alloys containing tungsten and molybdenum. The alloy gave the analysis: 88.18 per cent Cu, 3.22 per cent Cr, 1.35 per cent Fe, 2.38 per cent C, and 4.13 per cent gangue. The specific gravity was 8.3.

Binet de Jassonnix** stated that chromium dissolved in copper to the extent of about 1.6 per cent, but that on cooling, the chromium separated in a very finely divided condition.

G. Hindrichs†† made a more extensive study of the alloys of chromium and copper. He decided that the freezing point of copper was lowered about eight degrees by the addition of 0.5 per cent of chromium and that the maximum solubility of chromium in copper was 0.5 per cent. He, likewise, found that the freezing point of chromium was lowered from 1550 to about 1470 degrees C. by the addition of 5 per cent of copper and considered that the maximum solubility of copper in chromium was not over 5 per cent. From his researches it seems that there are two eutectic

*David H. Browne, C. A., Vol. 4, p. 41, 1910.

†H. Moissan, Compt. rend., Vol. 119, p. 185, 1894.

H. Moissan, Compt. rend., Vol. 122, p. 1302, 1896.

‡H. Goldschmidt, Liebigs Ann., Vol. 301, p. 25, 1898.

¶Thermit Carbon-Free Metals, Pamphlet No. 20, 2nd ed., p. 23.

§Hamilton and Smith, Jour. Am. Chem. Soc., Vol. 23, p. 151, 1901.

**Binet de Jassonnix, Compt. rend., Vol. 144, p. 915, 1907.

††G. Hindrichs, Z. anorg. Chem., Vol. 59, p. 414, 1908.

points in the chromium-copper freezing point curve. His chromium-copper diagram has been reproduced in Fig. 67. He pointed out some of the difficulties in working with chromium or chromium-copper alloys. Chromium remains viscous after melting, attacks the crucibles, insulating tubes, etc., and has a strong tendency to oxidize. He was not able to get chromium and copper to separate into two well-defined layers.

From the foregoing paragraphs it may be seen that the different investigators have placed the solubility of chromium in copper at 0.5, 1.6, 3.22, and 10 per cents.

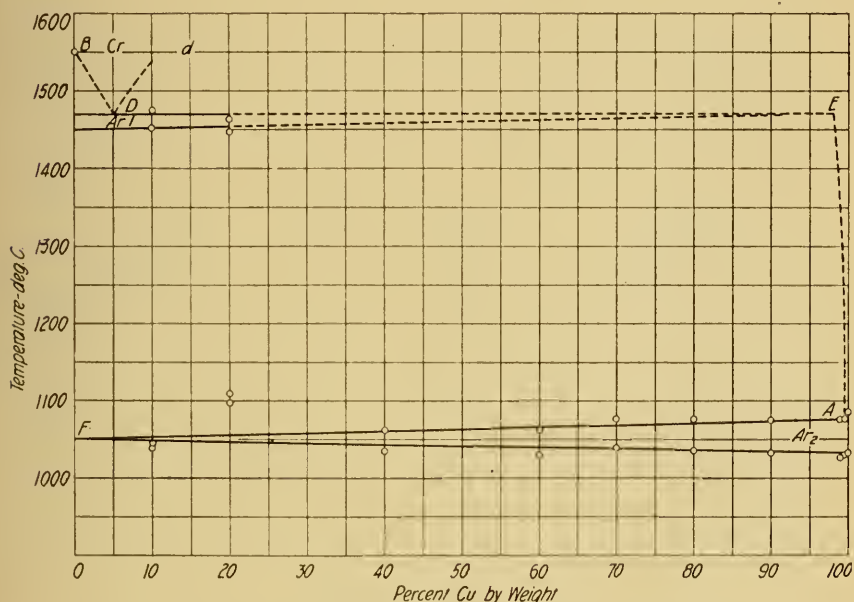


FIG. 67. COPPER-CHROMIUM EQUILIBRIUM DIAGRAM AFTER HINDRICHS.

3. *Chromium-Nickel Alloys.*—G. Voss* has investigated the alloys of chromium and nickel. His chromium-nickel diagram has been reproduced in Fig. 68. At the time he published his paper he stated that he was unable to find any published literature on the subject and at the present time he seems to be the only one who has published his researches. However, there have been extensive researches in the development of such alloys as "Nichrome", but these have been conducted in commercial laboratories and the re-

*G. Voss, Z. anorg. Chem., Vol. 57, p. 34, 1908.

sults have not been published. Voss showed that the system consists of two series of solid solutions with a minimum freezing point of about 42 per cent of nickel. From a microscopic examination he concluded that the point represented a true eutectic, although he was not able to demonstrate the presence of a eutectic structure on either side of this point. He assumed the presence of a solution gap (*Mischungslücke*). On the other hand Guertler* concluded that the structure represented a condition of unstable equilibrium, which he termed a psuedoeutectic produced by the extreme viscosity of the chromium.

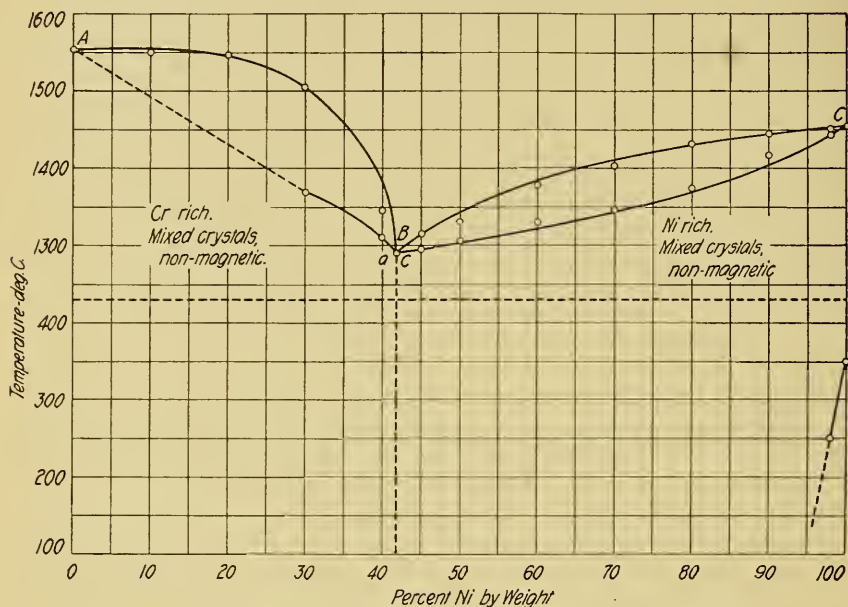


FIG. 68. NICKEL-CHROMIUM EQUILIBRIUM DIAGRAM AFTER VOSS.

To summarize, the binary alloys belong to three different classes: copper-nickel representing those which have continuous freezing-point curves, chromium-nickel representing those which have two series of solid solutions and a minimum freezing point, and chromium-copper those which have two eutectic points. The freezing-point curves for copper-nickel and for chromium-nickel seem to be pretty well established, but the exact location of a large part of the freezing-point curve for chromium-copper is unknown.

*Guertler, *Metallographie*, Vol. 1. Part I, p. 209, 1912.

4. *Ternary Alloys*.—The ternary alloys of chromium-copper-nickel have not been described, so far as is known to the writers, but four ternary systems which are somewhat closely related have been reported: copper-nickel-zinc by Victor E. Tafel,* copper-iron-nickel by R. Vogel,† copper-manganese-nickel by N. Parravano,‡ and cobalt-copper-nickel by Wahlert.§

*Victor E. Tafel, *Metallurgie*, Vol. 5, p. 413, 1908.

†R. Vogel, *Z. anorg. Chem.*, Vol. 67, p. 1, 1910.

‡N. Parravano, *Inter. Z. Metallographie*, Vol. 4, p. 171, 1913.

§Wahlert, *Oester. Z. Berg. Hüttenw.*, Vol. 62, pp. 341-6, 357-61, 374-8, 392-5, and 406-10, 1914. *C.A.*, Vol. 8, p. 3549, 1914.

- Bulletin No. 1.* Tests of Reinforced Concrete Beams, by Arthur N. Talbot. 1904. *None available.*
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- Bulletin No. 2.* Tests of High-Speed Tool Steels on Cast Iron, by L. P. Breckenridge and Henry B. Dirks. 1905. *None available.*
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- Circular No. 3.* Fuel Tests with Illinois Coal (Compiled from tests made by the Technologic Branch of the U. S. G. S., at the St. Louis, Mo., Fuel Testing Plan, 1904-1907), by L. P. Breckenridge and Paul Diserens. 1909. *None available.*
- Bulletin No. 3.* The Engineering Experiment Station of the University of Illinois, by L. P. Breckenridge. 1906. *None available.*
- Bulletin No. 4.* Tests of Reinforced Concrete Beams, Series of 1905, by Arthur N. Talbot. 1906. *Forty-five cents.*
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- Bulletin No. 9.* An Extension of the Dewey Decimal System of Classification Applied to the Engineering Industries, by L. P. Breckenridge and G. A. Goodenough. 1906. Revised Edition 1912. *Fifty cents.*
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- Bulletin No. 14.* Tests of Reinforced Concrete Beams, Series of 1906, by Arthur N. Talbot. 1907. *None available.*
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UNIVERSITY OF ILLINOIS BULLETIN

ISSUED WEEKLY

Vol. XIV

JANUARY 1, 1917

No. 18

[Entered as second-class matter Dec. 11, 1912, at the Post Office at Urbana, Ill., under the Act of Aug. 24, 1912.]

THE EMBRITTLING ACTION OF SODIUM HYDROXIDE ON SOFT STEEL

BY
S. W. PARR



BULLETIN No. 94

ENGINEERING EXPERIMENT STATION

PUBLISHED BY THE UNIVERSITY OF ILLINOIS, URBANA

PRICE: THIRTY CENTS

EUROPEAN AGENT

CHAPMAN & HALL, LTD., LONDON

THE Engineering Experiment Station was established by act of the Board of Trustees, December 8, 1903. It is the purpose of the Station to carry on investigations along various lines of engineering and to study problems of importance to professional engineers and to the manufacturing, railway, mining, constructional, and industrial interests of the State.

The control of the Engineering Experiment Station is vested in the heads of the several departments of the College of Engineering. These constitute the Station Staff and, with the Director, determine the character of the investigations to be undertaken. The work is carried on under the supervision of the Staff, sometimes by research fellows as graduate work, sometimes by members of the instructional staff of the College of Engineering, but more frequently by investigators belonging to the Station corps.

The results of these investigations are published in the form of bulletins, which record mostly the experiments of the Station's own staff of investigators. There will also be issued from time to time, in the form of circulars, compilations giving the results of the experiments of engineers, industrial works, technical institutions, and governmental testing departments.

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ENGINEERING EXPERIMENT STATION

BULLETIN No. 94

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PROFESSOR OF APPLIED CHEMISTRY

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THE EMBRITTLING ACTION OF SODIUM HYDROXIDE ON SOFT STEEL

I. THE EMBRITTLEMENT OF STEEL BY CAUSTIC SODA

1. *Introduction.*—Early in the year 1912 a certain boiler distress which had occurred at the University of Illinois brought to a focus the more or less controverted point as to whether the method of construction of the boilers in question, the material used, or chemical action was at fault. After numerous conferences with the representatives of the Babcock and Wilcox Company and the presentation by them of a mass of facts and opinions relating to similar difficulties elsewhere, it was agreed that there was sufficient evidence at hand practically to preclude the theory of faulty material, and that the subject was important enough to deserve the attention of a special committee to formulate a method of procedure for carrying out an extended investigation of the difficulty.

In consequence a committee was appointed by the Director of the Engineering Experiment Station as indicated by the following letter:

URBANA, ILLINOIS,
January 28, 1913.

PROFESSOR C. R. RICHARDS, Chairman; PROFESSOR E. J. BERG, PROFESSOR
A. N. TALBOT, PROFESSOR J. M. WHITE, PROFESSOR S. W. PARR, PRO-
FESSOR H. F. MOORE:

GENTLEMEN:—

At a meeting of the Station Staff held January 27 a statement was submitted by the Director outlining a problem in boiler maintenance which had been formally referred to the Station. The question raised concerns the cracking of plates in boilers which has been observed in a considerable number of instances in three different localities in the Mississippi valley. In all the cases reported the boilers were fed with artesian well water.

The undersigned reported that in the process of developing the question there had been some correspondence and that several of those interested in the matter at the University had two meetings with Mr. I. Harter, Jr., of the Babcock and Wilcox Company. At my request Professor Richards has formulated a statement, dated January 23, 1913, covering the investigation that should be made, which statement supplemented by briefs submitted by Professor Talbot and Professor Parr were presented in full at the meeting of the Station Staff. It was unanimously agreed that you should be appointed a Committee to have charge of the

proposed investigation, and the purpose of this letter is formally to notify you of your appointment upon such a Committee. I assume that the facts involved are already known to you and that the initiative in the further development of the problem will rest with the Chairman of the Committee.

May I add an expression of my belief that the problem which has been outlined is one of unusual importance and that its solution will justify the bestowal of generous attention upon the part of each member of the Committee.

Very truly yours,

W. F. M. GOSS,

DIRECTOR OF THE ENGINEERING EXPERIMENT STATION.

As a result of the conference of this committee it was agreed that certain of the more evident phases of the topic were of a chemical character and that these should be taken up by the department of chemistry. Dr. William Hirschkind of the division of Industrial Chemistry was detailed under the direction of Professor S. W. Parr to begin the work. His first report under date of April 15th, 1913, was supplemented by a more complete report covering all the phases of his investigation up to June 15th, 1913.

Following Dr. Hirschkind, the work was taken up again in March of the next year by Dr. Paul D. Merica and continued until July, 1914.

Before presenting the results of these investigations some of the preliminary facts are discussed which have led up to the work and to some extent have controlled or indicated the line of chemical investigation to be followed.

2. *A New Type of Water Supply.*—The water works at Urbana, Illinois, installed in 1884, has its source of supply in a gravel and sand stratum about 165 feet below the surface. This plant has the distinction of being the first to bring this particular type of water into service.

The character of the water is unique in that it is almost free from sulphates and has from sixty to seventy parts per million of free sodium carbonate. Chemical analysis shows it to contain the seemingly incompatible mixture of approximately four hundred parts per million of the carbonates of calcium, magnesium iron, and the sixty to seventy parts of free sodium carbonate first mentioned. It should be remembered, however, that in the natural water these properties are all present in the "half-bound" or bicarbonate form and, hence, are soluble. The water, therefore, not only has no permanent hardness

but is designated as having "negative hardness" in that it is self-purging and forms no scale whatever within the boiler.

In the year 1895, and following, a series of studies was in progress in the division of Industrial Chemistry at the University of Illinois which sought to follow the changes and interactions going on in the water within a steam boiler, under the existing conditions of temperature and pressure, for the purpose of arriving at an understanding of the processes involved in the formation of scale, the treatment which would be indicated for its removal, the causes of corrosion, and other factors. Two interesting facts developed which were not fully appreciated at the time.

First, because of the opportunity placed at our disposal for studying the water-supplies of certain railroads, especially the Illinois Central, the Big Four between Indianapolis and Peoria, and the Chicago and Alton, a rather extended zone for this particular type of water was indicated and fairly definitely outlined.*

Second. In waters of this type, after a few days of use in stationary boilers where the blow-off is naturally only partial and periodic, the sodium bicarbonate, $\text{Na}_2\text{H}_2(\text{CO}_3)_2$, of the raw water did not stop in its process of decomposition by heat



but went further; part of the sodium carbonate hydrolyzed and became sodium hydroxide according to the reaction:



The residual water within the boiler became an active reagent for the precipitation of scale-forming material the moment it came in contact with the fresh incoming water, thus indicating why such waters are self purging and develop no scale whatever within the boiler. Further significance which might attach to the presence of caustic soda in the boilers did not appear at that time.

Within the last few years a series of phenomena has developed which seems to have a direct interest in connection with the data of former years.

In addition to the distribution of such waters as described in the article referred to in the Journal of the American Chemical Society, the Illinois State Water Survey has found waters of this type to be

*Journal Am. Chem. Soc., vol. 28, p. 640, 1906.

much more widely distributed than formerly when the main supplies came from shallow wells. The local area may be roughly indicated* by drawing a line from a point somewhere between Paxton and Gilman on the Illinois Central Railroad, proceeding westward to include Normal, thence southward through the center of Bloomington somewhere between the C. & A. Junction and the pumping station of the Big Four at the foot of Center Street, thence south and east to include Bement, Tolono, Philo, on to Veedersburg, Indiana, and thence westward again to include Hoopeston and Paxton, the starting point.

COMPOSITION OF ALKALINE WATERS IN THE URBANA-CHAMPAIGN
DISTRICT, SHOWING THE HYPOTHETICAL COMBINATIONS IN
GRAINS PER UNITED STATES GALLON¹

Location	Urbana	Hoopeston	Paxton	Normal	Bloomington	Bement	Tolono
Depth of Well (in Feet) .	160	350	120	180	174	150	146
Supply	U. of I.	City	City	City	Big 4	City	City
Potassium Carbonate		.09					
Potassium Nitrate	.05					.04	
Potassium Chloride	.24	.12		.89		1.78	.32
Potassium Sulphate	.18	.08					
Sodium Nitrate			.07		.02		
Sodium Chloride			.26	.16	.77	1.98	.33
Sodium Sulphate			.39	.18	3.60	.37	.27
Sodium Carbonate	5.09	3.96	6.56	12.06	8.15	4.53	12.61
Ammonium Carbonate	.71	.09		.60		.32	1.59
Magnesium Carbonate	6.40	5.95	6.98	4.70	9.01	7.63	7.33
Calcium Carbonate	8.40	7.60	10.71	7.30	13.07	9.02	12.93
Iron Carbonate	.21	.19		.12	.01	.26	.22
Alumina	.05	.14		.05		.01	.23
Silica	1.47	.81	1.30	.81		.79	1.34
Bases	.00	.01		.15		.11	.06
Undetermined					1.57		
Total	22.80	19.04	26.27	27.02	36.20	26.82	37.23

¹ Data from the Illinois State Water Survey.

According to Bulletin No. 4 of the State Water Survey, pp. 28 and 29, another marked area is found in the Fox River Valley in the northern part of the state which includes Kane and Henry counties and the region about DeKalb.

3. *Boiler Difficulties.*—It so happens that coincident with the development of these areas, certain characteristic boiler troubles have appeared which have given no little concern to boiler users and makers alike. The details of boiler inspection reports show a decided increase of boiler faulting in areas where the water supplied to the boilers is

*Journal Am. Chem. Soc., vol. 28, p. 640, 1906.

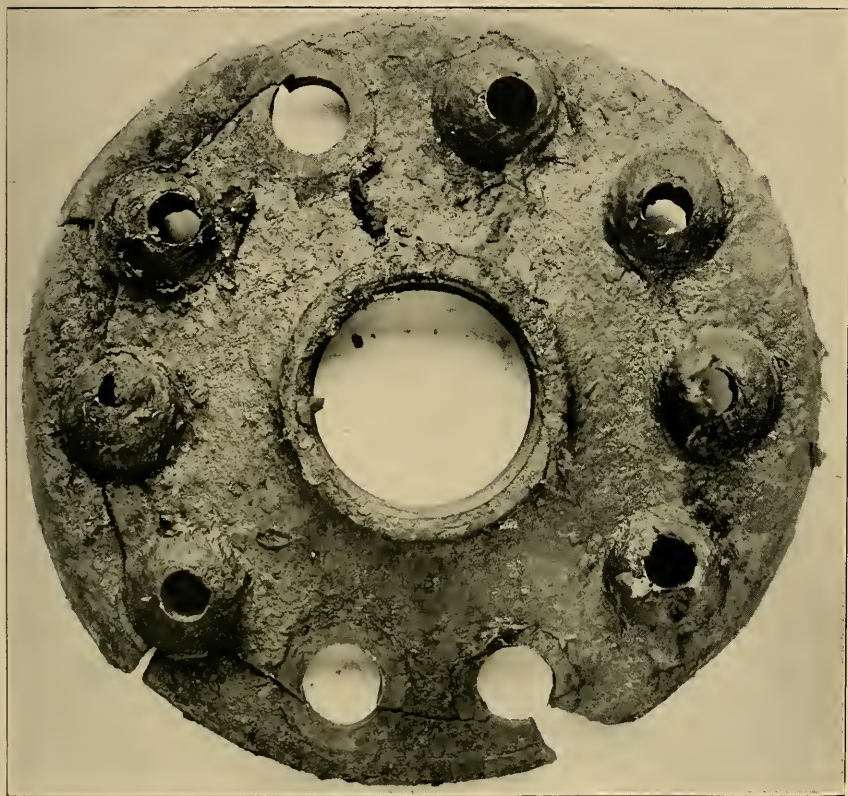


FIG. 1. A BLOW-OFF FLANGE OF A 260 H. P. STERLING BOILER

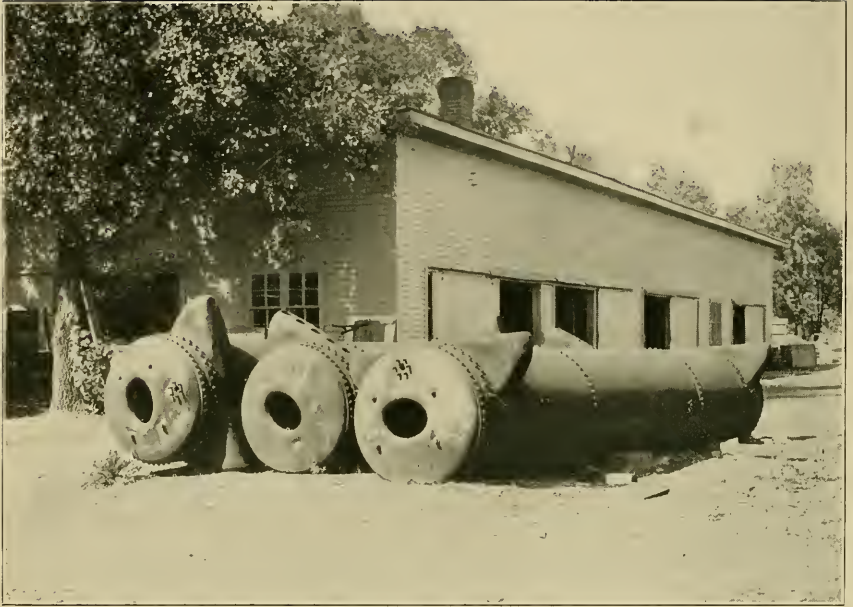


FIG. 2. PHOTOGRAPH OF DRUMS REMOVED, SUMMER OF 1915

of the character described. Without going into the various items, it is sufficient for purposes of this discussion to note that the number of boiler explosions occurring in the last twenty years and seemingly referable to the same cause, whatever that may be, have numbered twelve in the northern area above outlined and eleven in the southern.

The boiler distress is described as showing first in the form of fine cracks which develop and may proceed from the rivet holes to the surrounding plate, or from rivet hole to rivet hole, or from rivet holes to the edge of the plate where there is no tensile stress whatever. The development of these cracks occurs always below the water-line, and in the University of Illinois boilers, always in conjunction with a leak or other condition which promotes a concentration of the soluble material to the saturation point. Some of the characteristics of these cracks are shown in the accompanying illustration (Fig. 1) of a blow-off flange of a 260 H. P. Sterling boiler. Leakage had developed about this flange to an extent which made it desirable to remove the flange for refitting. The first attempt to drive out a rivet revealed the condition which is shown in its full development in the picture. Attention is called to the fact that the leakage area is confined to the outer edge of the flange; no leakage occurred at the center. Cracking is in evidence from rivet hole to rivet hole in every case and similarly every rivet hole has a crack extending outward to the edge of the flange.

4. *Purpose of these Studies.*—The purpose of this investigation has been to determine the embrittling effect of certain chemicals when brought into contact with iron and steel in such a manner as to set up a chemical reaction. It is not an attempt to explain the cause for boiler failures in these areas. Though it may throw some light upon one of the factors in the case which may be the predominating factor, emphasis should be laid upon the fact that much more remains to be done before a complete and adequate explanation can be offered. For example, the first twelve or fifteen years of use of this water in the University boilers was without noticeable effect on the boiler plates. In 1915 four drums in a new equipment of two 500 H.P. B & W boilers, which had been in use only three years, were so badly cracked that it was necessary to remove them. The old plant which was used for heating, however, operated only at less than 100 pounds pressure, and the use of the boilers was intermittent. In the new plant carrying both the heating and the power load of the University the service has been continuous, shutting down only for repairs. A constant

heavy overload has been maintained and the pressure carried at approximately 140 pounds.

Temperature, concentration, and continuity may, therefore, be important factors. Moreover, the soluble salts remaining in the water from these two particular districts, at least, consist almost entirely of sodium carbonate and hydroxide with only a minimum of sulphate and chloride present. The inhibitive effect or perhaps simply the diluting effect of larger percentages of sulphates and chlorides may be a determining feature.

It is evident, therefore, that many factors enter into the problem, and it has seemed wise to confine our attention to one factor only with the definite understanding that the immediate objective has been a contribution to our knowledge of the embrittling process and the conditions under which it may be brought about.

The first report presented herewith (Chapter II) is that of Dr. Hirschkind, and since he has compiled a valuable list of the more important references, this part of his paper is given entire in Appendix B, together with additions of more recent date.

It will be noticed that Dr. Hirschkind's line of investigation consisted, in the main, of a study of the change in potential of iron which had been subjected to the action of nascent hydrogen. The first pressures used were normal; hence the temperatures were generally confined to from 100 degrees to 120 degrees Centigrade.

In Dr. Merica's work (Chapters III, IV, and V) the action of caustic solutions was extended to include reactions under pressure and, consequently, at higher temperatures. Other effects than potential modifications were studied, as, for instance, the inhibiting effect of certain reagents.

5. *Summary.*—Certain conditions may be summarized as indicating the direction taken in carrying on the experimental work. These may be briefly enumerated. The peculiar cracking or faulting of boiler plates under consideration occurred:

- a. Below the water line.
- b. In connection with leakage at seams and rivet holes.
- c. Accompanied by an exterior accumulation of incrustation of strongly alkaline character.
- d. Where the water employed contained a very considerable amount of caustic soda (NaOH).

The experiments, as a result of this general agreement in the conditions, were directed toward determining the effect of caustic soda upon steel or the indirect effect of hydrogen resulting from such action. In general there seemed to be sufficient data for concluding that—

- a. Caustic soda of sufficient strength attacks iron with the generation of hydrogen. Indicating the reaction with the hydroxyl ion only, we would have $3\text{Fe} + 4\text{OH} = \text{Fe}_3\text{O}_4 + 4\text{H}$.
- b. Hydrogen in the nascent state, whether generated by alkali or acid, enters into the texture of the iron in a way to modify its physical properties.
- c. The hydrogen effect, at least in its first application, is transient, and after a period of rest or freedom from the hydrogen action, the iron reverts to its normal condition.
- d. Sodium carbonate is without action on iron; there is, therefore, no generation of hydrogen. The hydrolysis of sodium carbonate is directly dependent upon the temperature maintained, the withdrawal of the vapor of CO_2 , and the admission through the feed water of carbonated water. It is evident, therefore, that the chemical activity would vary with the ratio of hydrolyzation or the degree of concentration of the sodium hydroxide.
- e. Certain accompanying salts, as the chromates, have an inhibitive effect. Other salts, as sulphates, have at least the effect of acting as diluents. The limits of concentration for maintaining a condition which would be below the danger point have not been studied, but are features of the case which are of the utmost importance. A continuation of experiments along the lines suggested is in progress.

II. THE EFFECT OF CAUSTIC SODA ON IRON

6. *Introduction.*—The studies made up to the present time, which are given in outline in Appendix B, have proceeded on the assumption that hydrogen in the ionic form does not exist in metals. T. W. Richards in his investigations denies this assumption because

of the difficulty of demonstrating the presence of ions in metals. According to our present views concerning electricity, such a difficulty does not now exist. We have only to make the assumption that molecular hydrogen changes neither the potential nor the structure of the iron. Hydrogen ions, however, whether generated electrolytically or chemically, increase the potential of the iron and cause the so-called hydrogen brittleness.

Molecular hydrogen can be absorbed in large amounts by porous or finely divided iron; it does not change the electro-motive force.* On the other hand, iron treated with electrolytic hydrogen or with acids, that is, with hydrogen ions, becomes remarkably active and brittle.†

The experiment of Heyn,‡ in which iron becomes active and brittle when heated in a hydrogen atmosphere above 750 degrees and rapidly cooled down, has not found a satisfactory explanation so far, but according to our theory it may be explained without difficulty. At 750 degrees the iron begins to send out electrons which ionize the surrounding gas; in this case hydrogen ions are formed which, taken up by the iron, cause activity and brittleness.

The normal water as used in the University boilers contains no caustic soda and far too little sodium carbonate to have any action whatever on the boiler plates. However, by hydrolysis and by concentration in certain localized places a point may readily be reached where chemical action occurs with the liberation of hydrogen.

7. *Experimental Work.*—The first series of experiments was directed toward the measuring of the iron potential in a solution. The difficulty in this process lies in finding a solution which does not change the iron to either the active or the passive side. The best solution would be a normal solution of FeSO_4 , such as was used by Richards and Behr, but the preparation and handling of such a solution so as to avoid all oxidation is very difficult. The simpler method as employed by Grave was, therefore, adopted; that is, the potential was measured in a 1/10 normal solution of KOH. This solution converts the iron slowly into the passive state. From the rate of change, however, we can determine the original state of the iron piece.

*Zeitschrift für physikalische Chemie, Stoichiometrie, und Verwandtschaftslehre, vol. 58, p. 301, 1907.

†Proceedings Royal Society, vol. 23, p. 186, 1875.

‡Stahl und Eisen, vol. 20, p. 837, 1900.

The iron used was steel wire which had been annealed to eliminate the effect of the previous stress. A long piece of wire was cut into uniform lengths of six inches, heated in a tube of Jena glass to 700 degrees, and slowly cooled. The surface of these wires was covered with an oxide layer, which was carefully removed before testing.

The method used for measuring the electromotive force was the Poggendorf compensation method with a Deprez D'Arsonval galvanometer as a zero instrument. The standard cell was of the Weston type. The normal element was Ostwald's Calomel normal electrode. Between the 1/10 normal KOH solution and the normal KCl solution of the normal electrode, was a saturated solution of KCl.

Two pieces of annealed steel wire were dipped in a solution of 1/10 normal KOH. The potential against the Calomel electrode and the change of the potential with the time was measured.

Sample No. 1				Sample No. 2			
Potential immediately after dipping into the solution							
0.32 Volt				0.34 Volt			
After 5 min.	0.3000		0.0085	0.3035		0.0140	
After 5 min.	0.2915		0.00931	0.2895		0.0085	
After 10 min.	0.2822		0.0044	0.2810		0.0097	
After 10 min.	0.2778		0.0198	0.2713		0.0113	
After 30 min.	0.2500			0.2600			

These figures show that the potential at the beginning is about 0.3 volts and drops in one hour to 0.2600 volts. This is very different from the potential of a piece of the same steel wire which is made active before measuring. The easiest way to convert a piece of iron into the active state is by cathodic polarization. Another method was

Sample No. 1		Sample No. 2	
Immediately after dipping in our solution.			
0.573 Volt		0.600 Volt	
After 5 min.	0.560 Volt	0.574 Volt	
After 1 hour	0.537 Volt	0.480 Volt	

tried in this experiment: Pieces of steel wire of exactly the same kind as described were treated for a few minutes with dilute sulphuric acid; that is, with a concentrated solution of hydrogen ions. Then they were carefully washed to remove the rest of the acid, and the electromotive force measured in 1/10 normal KOH.

The different electromotive behavior of these pieces when compared with the original is evident. It shows distinctly that the hydrogen ion taken up increases the potential very materially; furthermore, the rate of change is much smaller.

Now if iron or steel treated with caustic soda of a certain concentration also takes up hydrogen ions, the potential must be increased and the breaking strain lowered. In order to test this, experiments were carried on as follows:

Steel wires three inches long annealed in the manner described were treated in boiling caustic soda solution of different concentration. The caustic soda attacks even Jena glass so strongly that the tubes were seriously affected by the heating. Therefore, two Jena glass tubes fitting into each other were used. The inside tubes contained the steel wire and the caustic solution and were drawn out a little to prevent evaporation. The evaporation and the effect on the outside tubes could not be entirely prevented, but were greatly reduced. For our experiments four caustic soda solutions of different strengths were used, 10, 25, 50 and 70 per cent. Each tube contained two pieces of annealed steel wire three inches long.

The tubes were heated in an air bath to from 110 to 120 degrees for one week. Afterwards the tubes were opened and the wires taken out and carefully washed with water. The wires in the 10 per cent NaOH solution did not show any change on the surface, but the wires in the stronger solution became rough, and dark colored, and contained on the surface adherent particles of solid caustic which were hard to remove.

The elongation and breaking strain of the original and of the treated pieces were determined through the courtesy of Prof. H. F. Moore. The results are given in Table 1.

The treated pieces show a slow increase in the elongation and a very distinct lowering in the breaking strain. Between the pieces in the various concentrated solutions there is only a small difference. If the lowering of the breaking strain is due to the hydrogen taken up, then the treated pieces must show a different potential in a solution.

TABLE 1
BREAKING STRAIN AS AFFECTED BY DIFFERENT STRENGTHS OF CAUSTIC
SODA SOLUTIONS

SPECIMEN	Diameter	Length		Ultimate	Elongation in Entire Length Per Cent	Unit Stress at Ultimate lb. per sq. inch
		Before	After			
Pieces of steel	0.104	2.95	3.68	510	24.75	60000
	0.104	2.93	3.65	510	24.60	60000
Wire annealed	0.104	2.94	3.66	520	24.50	61200
	0.104	2.95	3.73	510	26.40	60000
Untreated	0.104	2.50	2.94	510		60000
No. 1 Steel wires treated in 70 per cent NaOH .	0.104	2.96	3.74	470	26.30	55400
	0.104	2.97	3.72	470	25.50	55400
No. 2 Wires treated in 50 per cent NaOH	0.104	2.96	3.75	480	26.70	56600
	0.104	2.98	3.74	470	25.50	55400
No. 3 Wires treated in 25 per cent NaOH	0.104	2.95	3.80	485	28.80	57100
	0.104	2.94	3.70	475	25.90	56000
No. 4 Wires treated in 10 per cent NaOH	0.104	2.97	3.74	480	25.90	56600
	0.104	2.91	3.55	480	22.00	56600

In this way the electromotive force of the different treated and untreated pieces was determined.

The pieces of steel wire were carefully cleaned with sandpaper and fixed with paraffin in small glass tubes so that only a small part of the wire with the broken end came into the solution. In order to have a good contact a little mercury was admitted into the glass tubing which was closed with paraffin against the liquid. The method of measuring was exactly the same as in the preliminary experiments.

The results are given in Tables 2 and 3. The first series was measured about two days after the wires were taken out of the solution, the second about five days after. The figures show undoubtedly that the treated pieces have a considerably higher potential than the untreated.

The potentials in the second table are not quite as high as those in the first, probably because of a loss of hydrogen resulting from the exposure to the open air for a few days.

In order to test the action of caustic soda upon iron under stress another series of experiments was carried on as follows:

Four pieces of steel wire three inches long and four pieces six inches long having one bend in the middle were boiled in caustic soda solutions of the same concentration as in the experiments described.

TABLE 2

VARIATION IN POTENTIAL DUE TO TREATMENT IN CAUSTIC SODA SOLUTIONS OF VARYING STRENGTHS. MEASURED TWO DAYS AFTER TAKING OUT OF SOLUTION

SPECIMEN	Time	Poten- tial in Volts	Time	Poten- tial in Volts	Time	Poten- tial in Volts	Time	Poten- tial in Volts	Time	Poten- tial in Volts	Time	Poten- tial in Volts
Untreated wire.	4.08	0.370	4.13	0.332	4.19	0.300	4.30	0.287	4.45	0.275	5.05	0.255
No. 1. Treated in 70 per cent NaOH.....	4.09	0.428	4.14	0.401	4.20	0.380	4.31	0.359	4.46	0.331	5.06	0.301
No. 2. Treated in 50 per cent NaOH.....	4.10	0.437	4.15	0.407	4.21	0.391	4.32	0.346	4.47	0.351	5.07	0.332
No. 3. Treated in 25 per cent NaOH.....	4.11	0.405	4.16	0.381	4.22	0.366	4.33	0.346	4.48	0.322	5.08	0.303
No. 4. Treated in 10 per cent NaOH.....	4.12	0.399	4.17	0.378	4.23	0.359	4.34	0.344	4.49	0.327	5.09	0.308

TABLE 3

VARIATION IN POTENTIAL DUE TO TREATMENT IN CAUSTIC SODA SOLUTIONS OF VARYING STRENGTHS. MEASURED FIVE DAYS AFTER TAKING OUT OF SOLUTION

SPECIMEN	Time	Poten- tial in Volts	Time	Poten- tial in Volts	Time	Poten- tial in Volts	Time	Poten- tial in Volts	Time	Poten- tial in Volts	Time	Poten- tial in Volts
Untreated .	11.11	0.376	11.16	0.343	11.21	0.329	12.00	0.285	2.50	0.240	Next Day	0.217
No. 1	11.12	0.419	11.17	0.400	11.22	0.349	12.01	0.338	2.51	0.279	Next Day	0.220
No. 2	11.13	0.400	11.18	0.379	11.23	0.369	12.02	0.321	2.52	0.277	Next Day	0.220
No. 3	11.14	0.389	11.19	0.376	11.24	0.356	12.03	0.323	2.53	0.274	Next Day	0.236
No. 4	11.15	0.376	11.20	0.358	11.25	0.346	12.04	0.280	2.54	0.234	Next Day	0.184

TABLE 4

MODIFICATION OF THE ULTIMATE STRENGTH OF IRON AFTER TREATMENT UNDER STRESS WITH CAUSTIC SODA SOLUTIONS OF VARYING STRENGTHS
THREE-INCH LENGTHS

SPECIMEN	Diam- eter	Length		Ultimate	Elongation in Entire Length Per Cent	Unit Stress at Ultimate, Lbs. per Sq. Inch
		Before	After			
Untreated.....	1.04	3.05	3.75	935	22.0	110000
Pieces.....	1.04	3.00	3.70	770	23.3	90700
No. 1. 70 per cent NaOH..	1.04	3.01	3.75	610	24.6	71800
No. 2. 50 per cent NaOH..	1.04	2.96	3.74	558	26.4	65700
No. 3. 25 per cent NaOH..	1.04	2.93	3.65	560	24.6	66000
No. 4. 10 per cent NaOH..	1.04	2.86	3.63	570	26.9	67000

TABLE 5
MODIFICATIONS OF THE ULTIMATE STRENGTH OF IRON AFTER TREATMENT UNDER
STRESS WITH CAUSTIC SODA SOLUTIONS OF VARYING
STRENGTHS. SIX-INCH LENGTHS

SPECIMEN	Diam- eter	Length		Ultimate	Elongation in Entire Length Per Cent	Unit Stress at Ultimate, Lbs. per Sq. Inch
		Before	After			
Untreated.....	1.04	6.09	6.86	710	12.65	83000
Pieces one bend.....	1.04	6.09	6.86	710	12.65	83000
No. 1. Spoiled.....	1.04	5.96	6.80	480	14.10	56500
No. 2. 50 per cent NaOH..	1.04	5.97	6.80	490	13.90	57700
No. 3. 25 per cent NaOH..	1.04	5.94	6.76	490	13.80	57700
No. 4. 10 per cent NaOH..	1.04	5.94	6.76	490	13.80	57700

The heating was done in an air bath of from 100 to 110 degrees for one week. Afterwards the breaking strain was determined. The results are shown in Tables 4 and 5.

The pieces show after treatment a small increase in the elongation, just as in the first series of experiments. The ultimate stress diminishes with the treatment, especially in the pieces which were bent before insertion in the electrolyte.

In studying the effect of hydrogen ions on iron under stress the electromotive force of the bent pieces after treatment was not determined. The potential of the straight pieces was first determined immediately after they were taken out of the solution and again two days later after testing in the machine. The results are given in Tables 6 and 7. In the measurements of Table 7, the broken end of the tested pieces was dipped into the solution, as in the tests the results of which are given in Tables 2 and 3; in both series 6 and 7 the pieces used were carefully cleaned with sandpaper.

These experiments show again that the treated pieces have a distinctly more active potential than the untreated ones. The results

TABLE 6
EFFECT OF POTENTIAL OF STRAIGHT TREATED PIECES IMMEDIATELY AFTER
TAKING OUT OF THE SOLUTION

SPECIMEN	Time Min.	Volts	Time Min.	Volts	Time Min.	Volts
No. 4 10 Per Cent NaOH ...	0	0.4000	10	0.379	30	0.370
No. 3 25 Per Cent NaOH ...	0	0.470	10	0.466	30	0.453
No. 2 50 Per Cent NaOH ...	0	0.541	10	0.540	30	0.540
No. 1 70 Per Cent NaOH ...	0	0.520	10	0.510	30	0.5

given in Table 5, where the bent pieces show a much greater lowering of the breaking strain, require more investigation before conclusions may be drawn from them.

TABLE 7
EFFECT OF POTENTIAL OF STRAIGHT TREATED PIECES AFTER DETERMINING
THE BREAKING STRAIN. BROKEN END DIPPING INTO SOLUTION

SPECIMEN	Time Min.	Volts	Time Min.	Volts	Time Min.	Volts	Time Min.	Volts	Time Min.	Volts
No. 4 10 Per Cent NaOH..	0	0.422	10	0.3852	40	0.3565	3 hrs.	0.315		
No. 3 25 Per Cent NaOH..	0	0.451	10	0.4300						
No. 2 50 Per Cent NaOH..	0	0.457	10	0.434	10	0.432	40 min.	0.391	3 hrs.	0.338
No. 1 70 Per Cent NaOH..	0	0.400	10	0.370	10	0.356	40 min.	0.323	3 hrs.	0.296

As a result of our investigations, we are justified in making the following statements:

1. Pieces of steel wire, heated to from 100 to 120 degrees in caustic soda solution of a higher concentration than 10 per cent in a sealed tube, show a small increase in their elongation and a considerable lowering of the breaking stress.

2. At the same time the potential of these wires changes distinctly to the active side, which fact cannot very well be explained otherwise than by the taking up of hydrogen.

3. The electromotive test of any kind of iron and steel may give very important information, and it may be worth while to work out the details of this specific application to a greater extent.

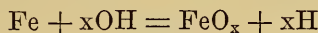
III. THE EMBRITTLING ACTION OF SODIUM HYDROXIDE ON SOFT STEEL

8. *Chemical Reactions Involved.*—It is well known * that at ordinary temperatures iron is passive or non-reactive to sodium hydroxide solutions having concentrations in the range of approximately 1/10 N to 2 N sodium hydroxide. If, however, the concentration of the hydroxide is increased, the passivity vanishes and a reaction of the sort described in the following may take place. This is shown by the electrolytic potential of iron toward solutions of sodium hydroxide or potassium hydroxide. A curve, Fig. 3, reproduced from Heyn and Bauer's work,† shows the Emf at ordinary temperatures of iron to

*Heyn and Bauer. Mitteilungen aus den Kgl. Material-prüfungsamt, 1908.

†Stahl und Eisen, 1908.

solutions of varying hydroxide concentration. It is seen that an anodic reaction of the type



becomes possible at hydroxide concentrations of about 5N-10N. At higher temperatures the reaction between sodium hydroxide and iron

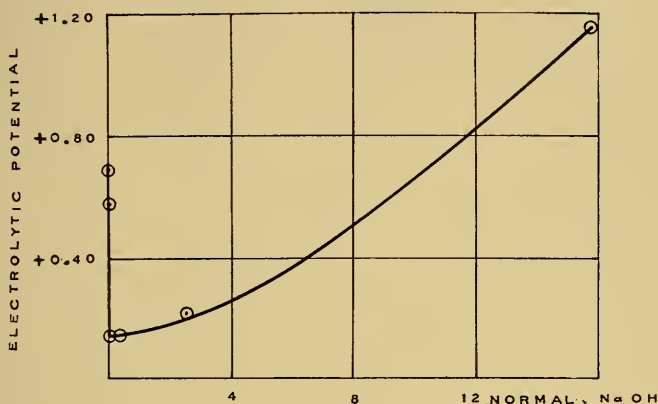


FIG. 3. THE ELECTROLYTIC POTENTIAL OF IRON TOWARD SODIUM HYDROXIDE SOLUTION

(ACCORDING TO HEYN AND BAUER)

Abcissas....Concentration of NaOH in mols per liter

Ordinates....Emf of Fe/NaOH/normal calomel electrode

is more rapid; Krassa * has shown that hydrogen gas is generated by the action of N sodium hydroxide on steel at from 150 to 200 degrees C. We may expect, therefore, to obtain hydrogen at temperatures above the ordinary by the action on steel of sodium hydroxide of higher concentrations.

9. *The Effect of Nascent Hydrogen upon the Physical Properties of Steel.*—The researches of Ledebur † into the so-called “pickling brittleness” have shown that very marked brittleness is produced in steel by treatment with dilute acid or by making it the cathode in an electrolytic cell. In both cases “nascent” hydrogen is generated on the surface of the steel and reacts with it, in some way not yet clear, to produce a material which is less tough than the original steel. The analogy of the two cases is now apparent, and it is not surprising

*Zeitschrift für Elektrochemie, p. 490, 1909.

†Stahl und Eisen, p. 681, 1887; p. 745, 1889.

to find that the action of sodium hydroxide on steel at higher temperatures, at which this reaction first proceeds with appreciable velocity, produces a brittleness comparable with the "pickling brittleness" and due to the same cause.

In Ledebur's study of this type of brittleness he found that it was not possible to show by the static tensile or compression test the deterioration which took place upon generating nascent hydrogen on steel. The various quantities measured, including elongation and reduction of area, remained practically unchanged. Some slight change was detected by a transverse static test, but when an alternating fatigue or impact test was used the deterioration was at once very definitely indicated; reductions in toughness of from 25 to 75 per cent were noted. In this investigation, therefore, it was decided not to rely upon a tensile test alone, but to include some form of impact and alternating stress test. This decision proved to be a wise one.

There is some reference in the literature to the effect that sodium hydroxide has a "destructive effect" on mild steel,* but detail data of the experiments are not at hand. Similarly an article has recently appeared by Andrews † on the same subject, in which, however, the description of his method of testing is not made clear. He states that a thin strip of soft steel became very "brittle" and "crystalline" after immersion for one week in sodium hydroxide at 100 degrees C., and he advances a theory to account for it. Further reference will be made to this article later.

Before describing the testing apparatus used, it may be well to give the reasons why small specimens were chosen for the tests. The chemical part of the problem requires a small specimen, since whatever effect the chemical may have on the material will be localized more or less at the surface, and in order to make the test sensitive it is necessary to have the ratio of the area of the section to the periphery small. This is done, doubtless, at the cost of accuracy, but for the reason stated it did not seem advisable to use standard dimensions either in the tensile test or in the impact bending test.

IV. DESCRIPTION OF THE TESTS, TESTING APPARATUS, AND MATERIAL TESTED

10. *The Tensile Test.*—The tensile tests were carried out on a 10,000 pound Olsen Machine; the same speed was used for all tests.

*Stromeyer Manchester Steam Users' Association, 1910.

†Transactions Faraday Society, March, 1914.



FIG. 4. IMPACT TESTING MACHINE

An automatic recording device was attached, and all the constants were read from the stress-strain diagram. It was considered possible that the form of this curve or the area under it might indicate any change in the quality of the material. The specimens were $2\frac{1}{4}$ inches long, and were turned down in the middle to a diameter of 0.150 inches for a length of $1\frac{1}{4}$ inches; thus a shoulder was left by which they were supported in the machine. This was considered better than threading the heads, as the thread would have been rendered useless by the corrosion due to the sodium hydroxide. It was intended to study the ultimate strength, the yield point, the elongation, and the area under the curve during the various treatments.

11. *The Impact Test.*—As there was no impact testing machine at hand which was small enough for these investigations, and time was an important factor, it was decided to build a special apparatus for the tests. Fortunately base and supports were already at hand in the form of a large drop testing machine, and it was only necessary to construct the pendulum, fasten it to the upright guides of the drop testing machine by means of wooden clamps, and bolt an anvil in place. An illustration of this machine as set up is presented as Fig. 4. The pendulum, consisting of a $1\frac{1}{2}$ in. by 4 in. by 6 in. steel block held by two half-inch bars, was supported in conical bearings from the side bars clamped to the fall-hammer guides. Its weight was 16.5 pounds (7.48 kg.). The striking face, which was hardened, had an angle of 45 degrees and was rounded with a radius of about $\frac{1}{8}$ inch. The anvil was a 2-inch steel block securely clamped to the base of the fall-hammer; it had an opening two inches wide with hardened anvil faces, rounded with a radius of about $\frac{3}{16}$ inch. The recording device was simply a pencil attached to the rotating axis of the pendulum which traced the record on a card tacked to a board secured at right

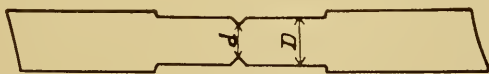


FIG. 5. FORM OF IMPACT TESTING SPECIMEN

angles to this axis. The height of fall of the center of inertia of the pendulum was taken at 24.6 inches (624 M). The round specimens, $4\frac{1}{2}$ inches long, were notched with a right angle tool in the lathe, and the material for about $\frac{1}{2}$ inch on either side of the notch turned down

to a definitely uniform diameter. The diameter at the bottom of the notch will hereafter be designated as d , the diameter of the specimen next to the notch as D . (For sketch showing shape of the specimens see Fig. 5.)

In the earlier tests the following procedure was adopted: The specimens were turned to the value of D and to within about 0.015 inch of the value of d as determined upon for the test. After the sodium hydroxide treatment, in which corrosion always took place, the notch was cut to the final value d , nothing further being done to

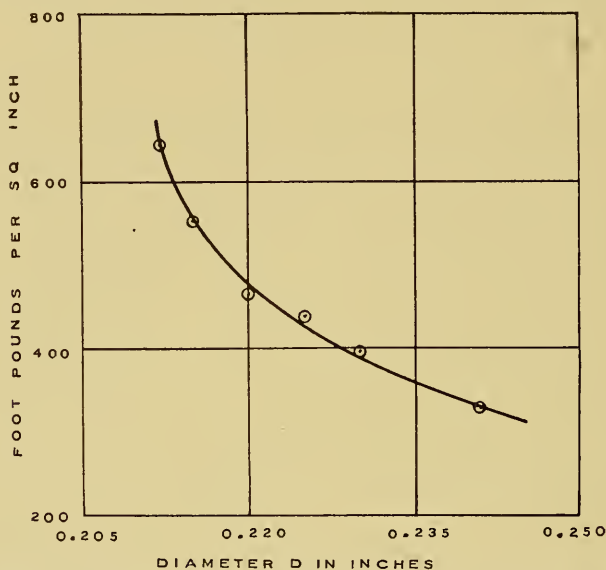


FIG. 6. EFFECT OF VARIATIONS IN THE DIAMETER D OF THE SPECIMEN (d CONSTANT) UPON THE SPECIFIC IMPACT WORK

Abscissas.... D in inches

Ordinates....Specific impact work in foot pounds per square inch

the specimen at D . Since, however, the specimen at this point was corroded down to a value of D , which was 0.002 to 0.004 inch less than the original dimension, values of the specific impact work (work measured relative to the area at bottom of notch) were obtained which were actually higher after the sodium hydroxide treatment than before, since the tests on the untreated material had been made on uncorroded specimens with the original value of D . Reference to Fig. 6 will show the effect of variation of D for a constant value of d

on the specific impact work. Fig. 6 also shows how great is the effect of variation of the value of D on the results obtained on the same material. After this had been noted in the experiment the procedure was adopted of turning down both at d and at D to the desired values *after* the treatment, thus securing the same conditions in all the tests. It may be mentioned that the same point was taken into consideration when the specimens were prepared for the repeated bending test, and the same order of operation observed.

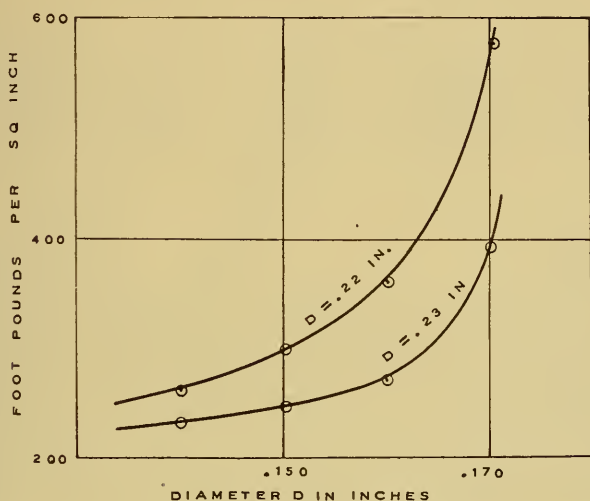


FIG. 7. EFFECT OF VARIATIONS IN THE DIAMETER d AT BOTTOM OF NOTCH (D CONSTANT) UPON THE SPECIFIC IMPACT WORK

Abcissas.... d in inches

Ordinates....Specific impact work in foot pounds per square inch

In tests run with any impact machine on notched specimens, there is some question as to the accuracy of the tests or accuracy of agreement, because there are two sources of error which are difficult to separate: (a) that which may be due to the machine, and (b) that due to the notching of the specimen. The latter is the more troublesome and difficult to avoid. With this machine an accuracy of agreement of values for the specific impact work of 10 to 15 per cent was generally obtained on specimens notched at the same time and under the same conditions. Variations of more than this were, however, observed when these conditions were not fulfilled.

Attention may be called to some of the results which were obtained with this machine in the early period of work on the dependence of the specific work of rupture on the diameters D and d . The curves showing these results are given in Figs. 6 and 7, in which the diameters, either D or d , are plotted in inches as abscissas, and the specific impact work in foot-lbs. per square inch, as ordinates. From a consideration of these curves it was decided to take for d the value 0.140 inches at which the variation of W , the specific impact work, with d was not so rapid.

12. *The Alternate Bending Test.*—A simple test of bending a thin wire back and forth in a vise gave to Ledebur the best indication of brittleness induced by pickling in acid. It was desired to apply, if possible, some more refined form of test to the present problem.

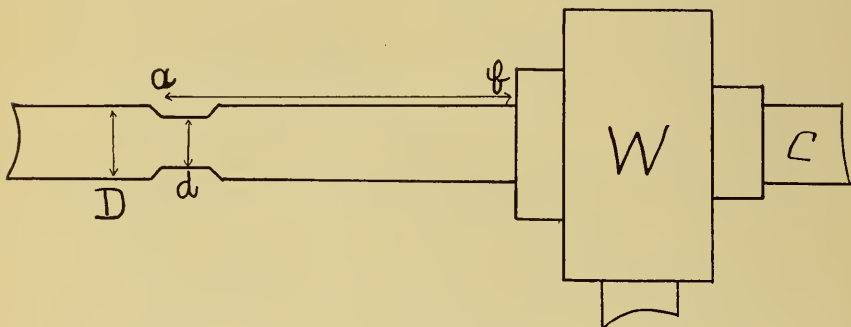


FIG. 8. ALTERNATE BENDING TEST SPECIMEN

Heyn has improved Ledebur's test by notching the specimen, and modern investigations on the "fatigue" of metals have brought out and developed many machines which apply a repeated bending stress. The one which seemed most suited to the present purpose was the White-Souther type. Although there was a White-Souther machine in the laboratory, it was considered best to use the arrangement of which an illustration is presented as Fig. 9. At one end the specimen is supported in a draw-chuck in a small lathe, and at the other in a bushing and ball-race, fitted in a frame. Through a universal joint this frame carries a spring balance by which the applied force is measured.

The shape of the specimen is shown by Fig. 8, the ball-race frame being indicated at W . The notch is cut with a right angle tool. As

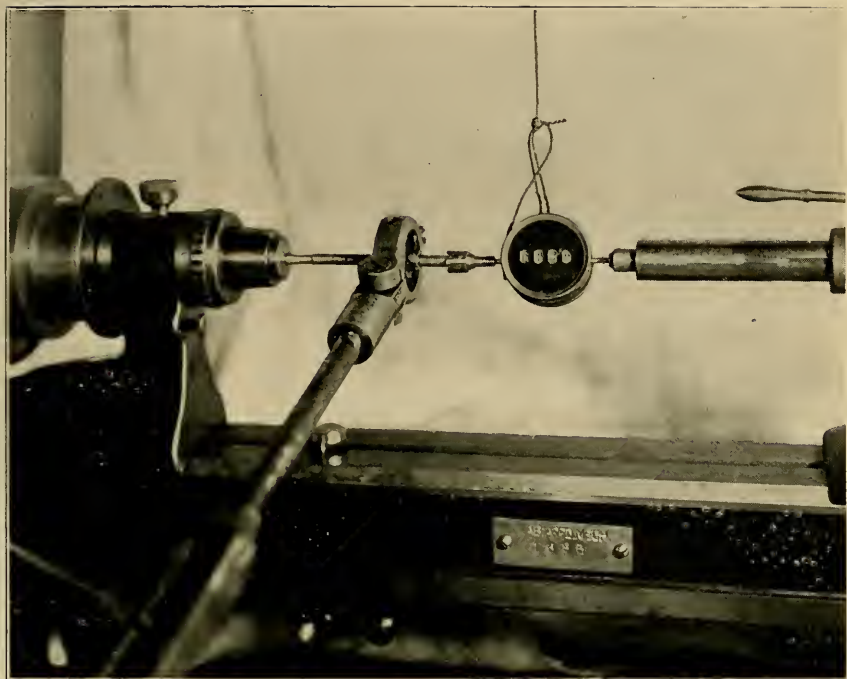


FIG. 9. ALTERNATE BENDING TESTING MACHINE

the fiber stresses in the notch depend on the dimensions of the specimen, in order to have comparable values it is only necessary to maintain as constants the force applied at W , and the lever arm of the force as measured from a , the end of the notch where the specimen breaks, to b , at which the force is applied. A rotation counter which registers the number of alternations required to rupture is attached to the free end of the specimen at C . The force and the lever arm were so chosen that about 300 alternations sufficed to rupture the

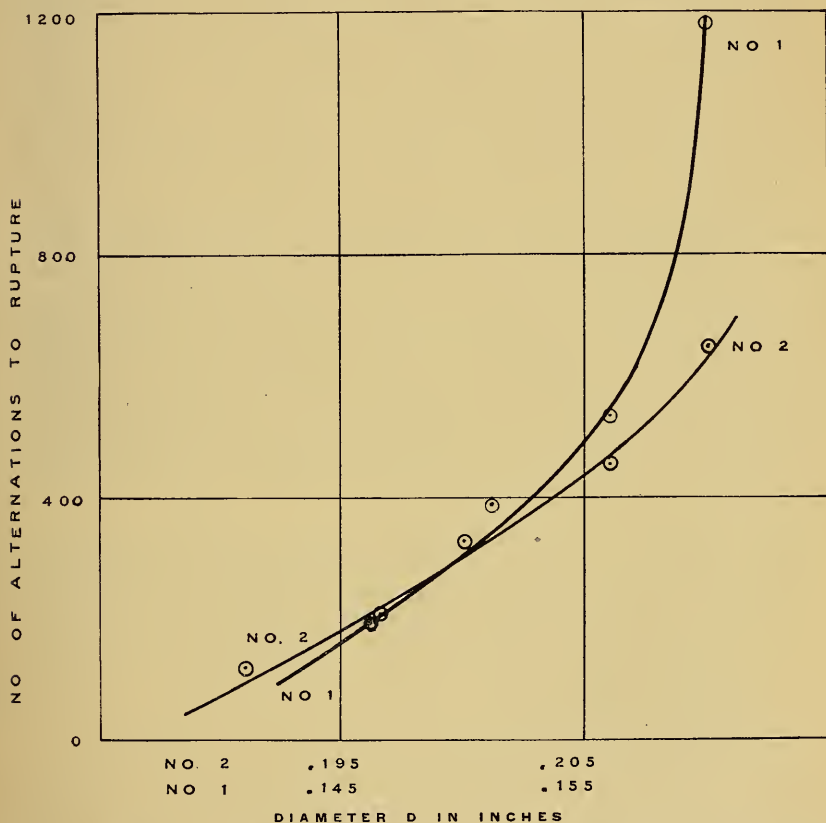


FIG. 10. THE ALTERNATE BENDING TEST. EFFECT OF VARIATIONS OF d (D AND M , THE BENDING MOMENT CONSTANT) ON THE NUMBER OF ALTERNATIONS TO RUPTURE

Abcissas.... d in inches

Ordinates....Number of alternations to rupture

specimen. All the tests were run at the speed of about 120 revolutions a minute.

It is recognized that present knowledge of the theory of repeated stress does not permit as definite conclusions from this test as from the tensile or even from the impact test. Nevertheless, this repeated bending test with a small number of revolutions has undoubtedly been successful in indicating brittleness in many cases where the tensile test failed, as for example, in the cases of phosphorus brittleness,* of pickling brittleness,† and of overheated steel.‡

As to the consistency of results, here again it was found that much depended upon the method of preparing the specimen. Reference to the curves in Fig. 10 shows that it was necessary to turn the specimen accurately. These curves were obtained in a series of tests on the untreated material by using a constant bending moment at a and a constant value for D , and by varying the diameter at the bottom of the notch d . Two ranges of values for d are shown, the values of the bending moment being constant. It is seen that for the lower value of d the rate of variation of the number of alternations to rupture (ordinates) is much greater than for the higher value. It was decided, therefore, to use the higher value for d (0.195) in the actual tests. In any series of tests on the same material under the same treatment, a maximum variation from the mean of about 5 to 10 per cent was usually found, although occasional variation of 50 to even 75 per cent would be found. Such exceptions were likely due to a lack of homogeneity in the material.

13. *Treatment of the Specimens with Sodium Hydroxide.*—It was decided to study the action of sodium hydroxide on steel at three temperatures, 100, 180 and 280 degrees C, and to use, for the most part, a concentration of 13.6 N sodium hydroxide or 38 per cent. In a few cases as noted in the tables a different concentration and different substances were used.

The treatment of the specimens at the lower temperature (100 degrees), even with the concentrated alkali, offered no difficulties, since it could be done in Jena glass tubes, and by putting a small funnel in the top of the tube the loss by evaporation could be minimized. At the higher temperatures, however, the glass would have

*Kommers, Proc. Int. Soc. Test. Mat., 1912.

†Stahl und Eisen, p. 681, 1887; p. 745, 1889.

‡Heyn, Handbuch der Materialienkunde, 1912.

been destroyed in a very short time, and some other material had to be sought, therefore, as a containing vessel for the solutions. Iron pipe was regarded as the simplest thing to use, but it was found impossible, even with the exercise of great care in cutting threads and with the use of asbestos and lead gaskets, to cap both ends securely enough to prevent the solution from evaporating completely in a short time. This is interesting in view of the fact that water can be held very easily under the same conditions, and it throws some light on the persistence of leakage at boiler seams when alkali is present. The next procedure after the solution and specimens had been put in was to weld up the tubes with the oxyacetylene torch. This was done by immersing the pipe during the welding in cold water to a point slightly above the level of the solution inside. It is again interesting to note that some of the welded pipes containing sodium hydroxide which were pressure tight upon the completion of the weld, after being heated for some days, developed leaks due to the action of the alkali. In general, however, this method of treatment was satisfactory.

The 100 degree temperature was obtained by means of a steam bath. The two higher temperatures were obtained by means of electrically heated air-baths which were constructed for this purpose by winding a 15-inch length of asbestos covered 6-inch iron pipe with nichrome wire, smearing it with a mixture of talcum and water glass, and putting it in a box of sand. In series with each of these two air-baths was a regulating resistance.

14. *Description of Steel Used.*—Owing to the short time allotted to this work, it was possible to carry out tests on only one material. This was $\frac{1}{4}$ -inch open-hearth wire obtained from the Kokomo Steel and Wire Company, and had the following composition:

C	0.18 per cent
P	0.024 per cent
S	0.045 per cent
Mn	0.42 per cent

Two batches of five hundred feet each were annealed in a large gas furnace in the forge laboratory of the University, and, unless otherwise mentioned, it was this material which was used in all the work.

It is to be noted that this steel is of a composition such as is specified for boiler plate. The significance of this will be seen in the following discussion.

To summarize, the specimens were prepared by turning down each to within about 0.015 inch of the diameter required in the tests, and were placed in a pipe which contained four or five of the samples for both the impact and bending tests and two for the tensile strength determinations. The solution was added, and the pipe sealed and put in the furnace for the required period. It was then removed, opened, and the specimens turned down to the correct dimensions and tested. Care was taken in this final turning not to heat the specimens, since it was thought that such a re-heating might affect the results. The results of the impact and of the repeated bending tests as presented in Table 8 are the averages of four or five tests; the values for tensile strength are the averages of two results.

TABLE 8

SERIES 1 OF TESTS SHOWING EFFECT OF ALKALI TREATMENT ON PHYSICAL PROPERTIES OF STEEL EXPRESSED IN PERCENTAGES OF THE CORRESPONDING VALUES FOR UNTREATED MATERIAL

SOLUTION	Time of Treatment in Days	Temperature of Treatment Degrees C	Tensile Test				Alternate Bending Test. No. of Alternations to Rupture*	Impact Test Specific Impact Work*	Hydrogen Evolution
			Yield Point Lbs. per Sq. In.*	Ultimate Tensile Strength Lbs. per Sq. In.*	Elongation in 1¼ Inches Per Cent*	Area under Stress-Strain Curve* Per Cent			
13.6 N NaOH..	17	100	116.0	102.0	100.0	112.0	85	92	
	30	100	101.0	102.0	83.0	92.0	110	108	
	10	180	110.0	100.5	98.0	110.0	64	...	Strong
	15	180	100.8	102.0	97.4	107.0	69	...	Strong
	15	180	103.0	98.5	82.0	88.0	82	...	Strong
	19	180	80.0	97.0	88.0	92.5	82	100	Strong
	23	180					88	100	Slight
	24	180					90	88	Strong
	12	180							Strong
	5	280	100.6	98.6	104.0	113.0	79	...	
	8	280	97.6	95.6	67.0	70.0	56	...	Strong
	10	280	99.6	100.1	97.0	110.0	57	83	Strong
	13	280	98.5	96.0	87.0	92.0	80	102	Strong
	15	280	89.0	94.4	98.0	104.0	75	104	None
	15	280	97.6	101.0	90.6	91.0	73	92	Slight
	15	280					79	104	Strong
	16	280	107.0	97.0	97.0	104.0	77	94	Strong
13.6 N NaOH + Na ₂ Cr ₂ O ₇ ..	15								Slight
	17	180	102.0	104.0	104.0	117.0	126	106	None
	19	180	116.0	105.0	101.0	116.0	105	102	None
	19	180							Yes
	17	280	96.5	101.0	87.0	94.5	119	86	None
	19	280	96.5	96.0	94.0	101.0	93	107	Slight
	20	280							None
Na ₂ CO ₃	23	180					104	109	Some
	..	280							Some
	16	280	101.0	100.0	91.0	97.0	131	104	None
	24	280							None
Boiler Scale.	19	180	106.0	101.0	94.0	104.0	91	94	
	23	180					90	103	Strong
	19	280							Slight
	19	280							None

*The results are expressed in terms of percentages of the values on original untreated material.

V. RESULTS OF THE PHYSICAL TESTS

Table 8 presents the results of the first series of tests; all values are expressed for convenience in terms of percentages of the corresponding values for untreated material. Column 9 indicates the amount of hydrogen set free during the action of the solution as evidenced by the amount of pressure developed. The material in the untreated state had a yield point at about 37,700 pounds, an ultimate strength of 55,900 pounds, and an elongation of 31 per cent on $1\frac{1}{2}$ inches. The area under the stress-strain curve was 1,190 foot-pounds per cubic inch. The specific impact work in the bending test when a deep notch ($d = 0.150$ inch) was used was 248 foot-pounds per square inch (5.36 m.-kg. per sq. cm.). No value can be given for the repeated bending test which would have any absolute significance, because of the difficulty of calculating the fiber stresses in the notch.

The results of the tensile test do not show any effect of the sodium hydroxide on the yield point or on the ultimate strength. The impression is gained from a study of the elongation that its value is reduced slightly by treatment in sodium hydroxide; this can also be said, although to a less degree, of the area under the curve. At the same time the reduction is not very marked, many of the values being reduced only by an amount easily within the experimental error.

Some of the results of the impact test of this series are without value and are omitted here as they were obtained before the procedure had been adopted of turning the specimen down both at d and at D after the treatment. The results given here show that the material did not deteriorate at 100 degrees and only slightly at higher temperatures.

When the results of the repeated bending test are considered, however, the effect of sodium hydroxide in embrittling is at once apparent; the number of alternations of stress withstood after treatment is, on an average, about 80 per cent of the values for the untreated material at the two higher temperatures. There are only two cases in which there is no decrease.

In order to confirm these results and at the same time to test the effect of variations in time and temperature on the values obtained, two more series of tests were run, the results being presented in Tables 9 and 10.

It is to be noted in all the tests that at both temperatures, 280 and 180 degrees C, the brittleness increased for about a week and

TABLE 9

SERIES 2 OF TESTS SHOWING THE EFFECT OF ALKALI TREATMENT ON PHYSICAL PROPERTIES OF STEEL

SOLUTION	Time of Treatment in Days	Temperature of Treatment Degrees C	Tensile Test				Alternate Bending Test. No. of Alternations to Rupture*	Impact Test. Specific Impact Work*	Hydrogen Evolution
			Yield Point Lbs. per Sq. In.*	Ultimate tensile Strength Lbs. per Sq. In.*	Elongation in 1¼ Inches Per Cent*	[Area] under Stress-Strain Curve Per Cent*			
13.6N NaOH..	8	100	96	101	96	104	95	113	
	30	100	...	...	...	...	90	87	
	8	180	115	100	102	110	93	76	Strong
	3	280	...	...	...	...	83	97	Strong
	8	280	98	97	104	109	88	97	Strong
13.N Na ₂ Cr ₂ O ₇	8	180	100	102	90	97	103	100	None
	8	280	94	102	88	96	96	122	Slight

*The results are expressed in terms of percentages of the values on original untreated material.

TABLE 10

SERIES 3 OF TESTS SHOWING EFFECT OF ALKALI TREATMENT ON PHYSICAL PROPERTIES OF STEEL

SOLUTION	Time of Treatment in Days	Temperature of Treatment Degrees C.	Alternate Bending Test. No. of Alternations to Rupture*	Impact Test. Specific Impact Work*	Hydrogen Evolution
13.6N NaOH	9	200	90	88	Strong
	14	200	106	89	Strong
	15	200	106	100	Strong
	14	200	106	100	
25N NaOH.....	15	200	82	86	Strong
N/10 NaOH.....	15	200	108	100	None
13.6N NaOH.....	9	200	106	104	None
13N Na ₂ Cr ₂ O ₇	14	200	100	89	None
25N NaOH.....	15	200	...	108	None
	15	200	...	93	None

*The results are expressed in terms of percentages of the values on original untreated material.

then decreased, the material recovering either partially (first series) or even wholly (second series) its original mechanical properties. This is indicated in the curves showing the results of the first and second series of repeated bending tests given in Fig. 11. Even more striking is the complete recovery noted in the third series. This was also noted by Andrews in his work. A tentative explanation of this rather curious phenomenon is presented in the following paragraph.

During the first period of the action of the alkali, before the oxide product of the corrosion has been formed, the hydrogen liberated by this action is actually given up in the nascent state on the iron

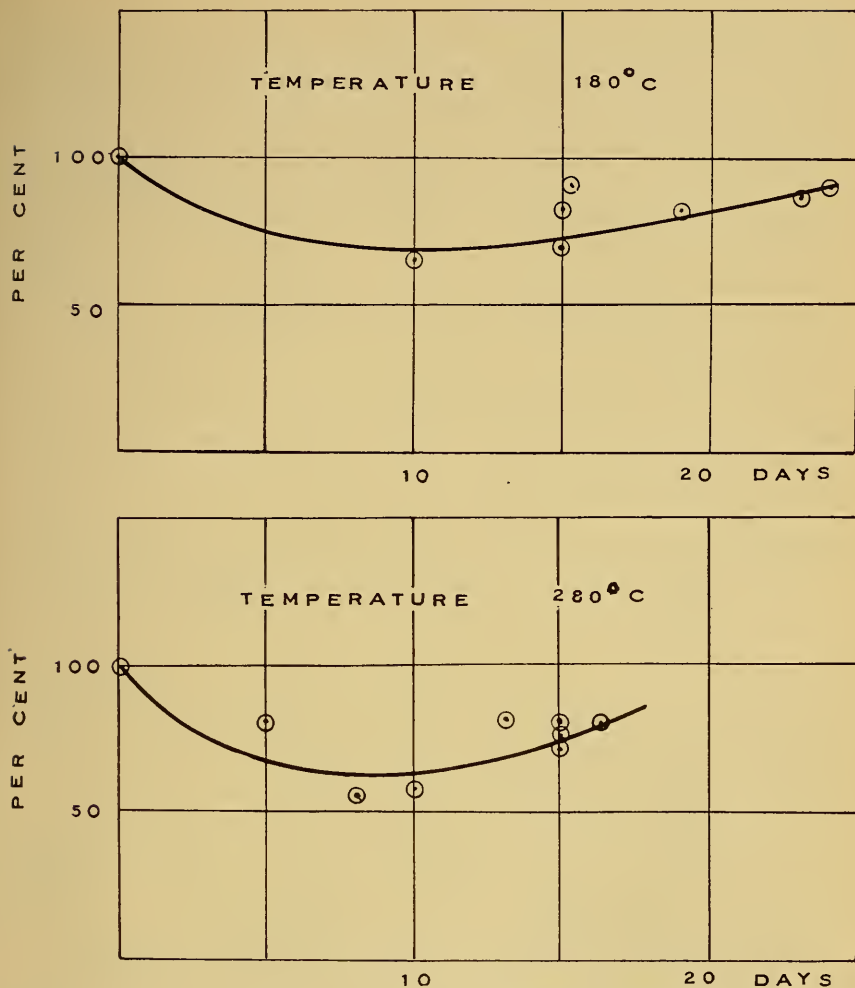


FIG. 11. EFFECT OF SODIUM HYDROXIDE TREATMENT UPON THE TOUGHNESS OF STEEL AS INDICATED BY THE ALTERNATE BENDING TEST. TEST SERIES I

Abcissas....Length of time of treatment in days

Ordinates....Number of alternations to rupture expressed as percentages of this value on original untreated material

and absorbed by it. Later the H_2 formed by the anode solution of the iron is given up at the cathode, which is now the very electro-negative oxide, and can come in contact with the iron itself only by secondary diffusion or after it has formed in the molecular state.

Such hydrogen has been shown to have little or no effect on the mechanical properties of steel. Therefore, as soon as the oxide has formed in a thick, dense layer over and near the iron, the embrittling action of the chemical agent ceases. It has also been shown by Heyn * and others that chemical (pickling) brittleness is relieved by heating for a short time at even such temperatures as 100 degrees C, the hydrogen evidently diffusing out of the material and leaving it in its original state. Therefore, after the embrittling action of the alkali

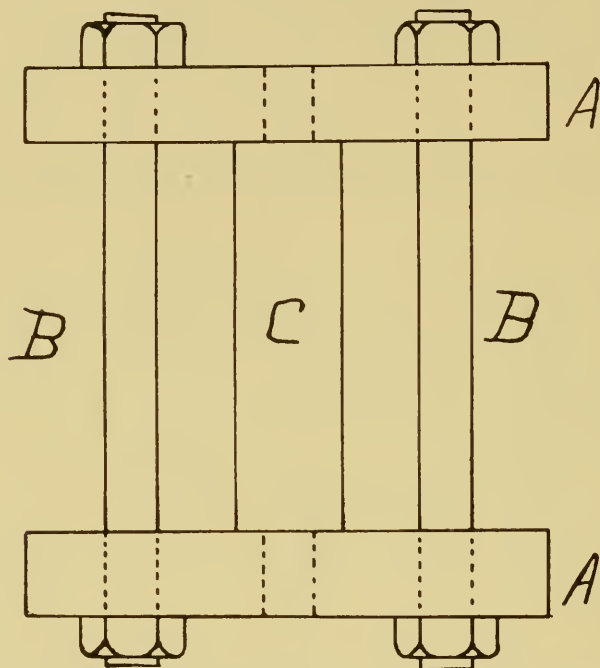


FIG. 12. FRAME FOR SUBJECTING STEEL TO CORROSION UNDER STRESS

has ceased, that brittleness which has already been induced is removed by the heating to which the material is further subjected.

The effect of the temperature of treatment on the results can be best seen in Table 12, from which it is evident that at the higher temperature the brittleness is developed in a shorter time than at the lower, and also the effect produced by the alkali is greater.

A survey of the results presented in the four tables shows, as would be expected, that the action of the alkali was more marked at

*Stahl und Eisen, vol. 20, p. 837, 1900.

the higher concentration of 25N than at 13N, and that in two weeks no effect at all was produced by the 1/10 normal alkali at 200 degrees C, although there was quite noticeable corrosion.

As it has been suggested by Stromeyer that steel in tension was more affected by treatment with sodium hydroxide when under stress than when without stress, and as such a fact would, under the circumstances, be of great significance, some tests were carried out to determine whether such an acceleration of the embrittling action could be noticed. Two specimens were put in a frame, a sketch of which is presented as Fig. 12. The specimens BB were then put in tension by screwing up the bolts, the amount of the stress being measured approximately with a strain gage. Since the specimens were notched it was impossible to calculate the fiber stresses at the notch; the average stress applied to the bottom of the notch was about 15,000 pounds. Three of these frames were then welded up with the solution in a 14-inch length of 4-inch pipe and heated for the desired period. It was found very difficult to make the weld in 4-inch pipe absolutely tight and free from pinhole leaks.

The results of these tests are presented in Table 11.

TABLE 11
EFFECT OF SODIUM HYDROXIDE TREATMENT ON SOFT STEEL UNDER STRESS

SOLUTION	Time of Treatment in Days	Temperature of Treatment Degrees C.	Alternate Bending Test. No. of Alternations to Rupture*	Impact Test. Specific Impact Work*	Hydrogen Evolution
25N NaOH	8	180	76	86	
25N NaOH	8(t)	180	95	65	
13N NaOH	14	190	97	..	Slight
13N NaOH	14(t)	190	110	..	

*The results are expressed in terms of percentages of the values on original untreated material.

(t) Specimens were in tension during treatment.

So far as can be seen from the results of the two runs made, material in tension shows, if anything, less tendency to undergo the embrittling attack of the alkali than material which is not. In view of the conflicting testimony on this point, it is probable that further investigations are needed.

Owing to the fact that the concentration of hydrogen is greatest at the surface and becomes less toward the center, the results of these tests probably do not give the proper evaluation of the amount of brittleness caused at the surface. Further, in turning down the

specimen after the treatment, about 15 to 20 per cent of the material which is most affected by the treatment is removed, so that the actual values over the whole alkali-treated section must have been still smaller by some fraction of the measured reduction in the values.

15. *Cause of the Embrittling Effect.*—The evidence seems to show that the embrittling effect of sodium hydroxide on steel is due to the evolution of hydrogen and the absorption by the steel of the hydrogen in the nascent state. However, there are two other possibilities which may be considered:

First, there is the possibility that the alkali actually works into the metal eating out the “amorphous inter-crystalline cement” which, being thermodynamically less stable than the crystals, would be first attacked. This would cause a brittleness akin to that caused by burning or overheating and fracture would be intercrystalline.

Second, Andrews claims that sodium hydroxide causes a “crystallization,” by which he means a coalescence of the smaller crystals into larger ones. This effect would also be the same as that caused by over-heating.

It is not believed by the author that sodium hydroxide acts in this latter way, or, at least, that the embrittling effect is due to any such action. It is, in the first place, difficult to see in just what manner the presence of sodium hydroxide could cause a coalescence to take place in the period of one week which it takes to develop this brittleness, at the comparatively low temperatures of 100 to 200 degrees, in material in which the coalescence is known to proceed at a scarcely appreciable rate at these temperatures.

It has been further shown that the material in contact with sodium hydroxide at higher temperatures undergoes a gradual recovery of the original mechanical properties, a phenomenon which has already been given an explanation, and which cannot possibly be reconciled with any theory making “recrystallization” responsible for this effect. There are presented herewith some photomicrographs to show that no such coalescence does actually take place. Besides the material used for the tests, some specimens of electrolytic iron were used in studying this question. Pieces of the material about one inch long were turned down to a uniform diameter and cut in two. One-half was now treated at any desired temperature with sodium hydroxide and the other held for comparison.

In order to create a favorable condition for crystallization all

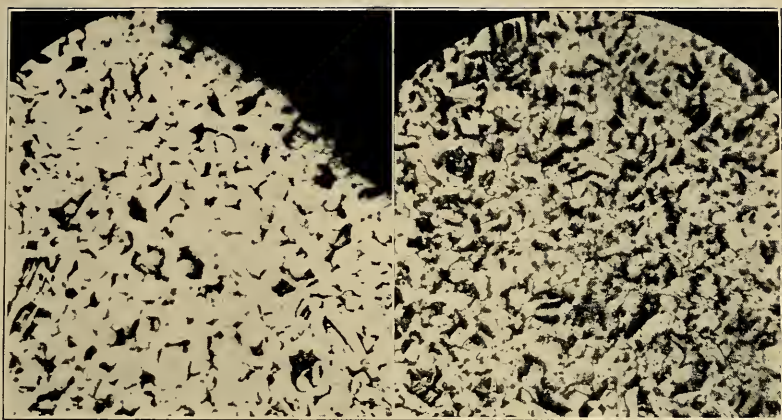


FIG. 13. PHOTOMICROGRAPHS OF SOFT STEEL, BEFORE AND AFTER
TREATMENT WITH SODIUM HYDROXIDE

of the electrolytic iron and some of the steel specimens were first heated above 900 degrees and cooled quickly in air, thus producing a fine, hence less stable, grain. After treatment a low melting alloy was cast around each specimen; in order that the structure at the edge where the alkali had acted might be observed, about 1 mm. was then ground off, and the specimen polished and etched with picric acid.

The specimens were examined and in no case was any increase in grain size noticed as a result of the action of the alkali (Table 12). Photomicrographs, Fig. 13, show the appearance of the steel used before and after treatment with sodium hydroxide at 200 degrees C for fifteen days.

TABLE 12

EFFECT OF ALKALI TREATMENT AT HIGH TEMPERATURE ON GRAIN SIZE OR CRYSTALLIZATION OF IRON AND STEEL

MATERIAL	Period of Treatment with Alkali	Temperature of Treatment	Solution
Steel annealed at 900 degrees and (1) cooled in furnace..... (2) cooled in air.....	21 15	280° C. 200° C.	13.6 N NaOH 13.6 N NaOH
Electrolytic Iron Remelted annealed in vacuum and cooled in air.....	15	200° C.	13.6 N NaOH

Although it is possible that an effect similar to the first mentioned may be present as an auxiliary cause, all the facts seem to show that absorbed nascent hydrogen is the principal cause of brittleness. It is impossible at present to say in just what state the hydrogen is present in the iron, whether as a solid solution, as a compound, or as a solid solution of that compound. It is, indeed, probable that even to detect analytically the small amounts of H_2 present would have been extremely difficult, and for that reason it was not attempted. That hydrogen is evolved during the action of alkali on the iron could be observed at all temperatures above 100 degrees. Upon opening the pipes containing the alkali, a gas was found under considerable pressure, which analyzed about 94 per cent H_2 .

A rather striking proof of the fact that some molecular change takes place in iron during the action of alkali is found in a study of its Emf toward any strength of solution, more conveniently N/10 sodium hydroxide, before and after treatment in concentrated alkali

at temperatures of 100 degrees and more. This matter was studied first by Dr. Hirschkind and later by the author, and it was found by both of us that after treatment the potential is considerably higher than before, and that it extends into the specimen from the surface

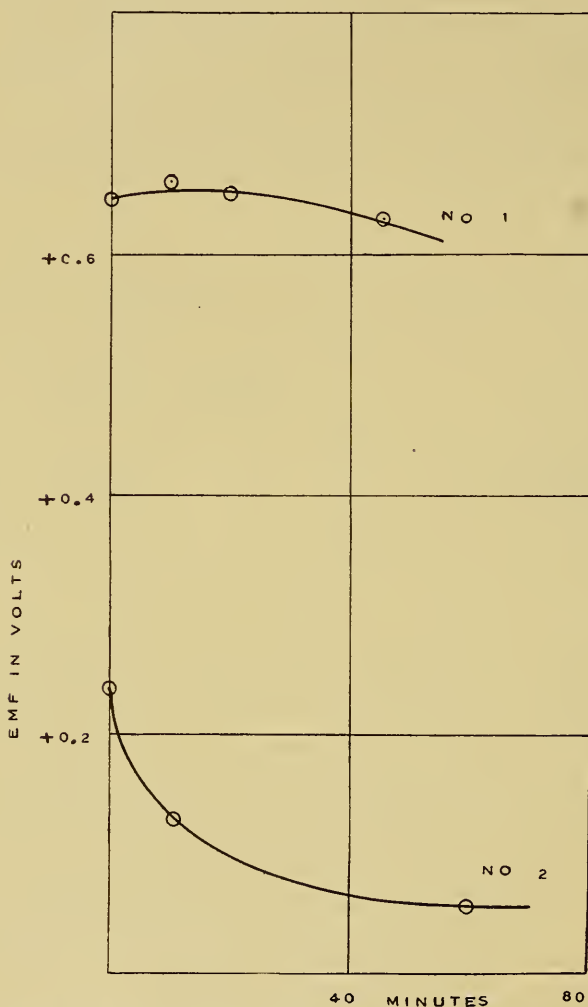


FIG. 14. EMF OF IRON TO N/10 NaOH BEFORE AND AFTER TREATMENT WITH CONCENTRATED NaOH

(Emf measured against the $\text{Hg/HgO} \frac{\text{N}}{10} \text{NaOH}$ electrode)

for a considerable distance; this last fact was proved by turning off from 0.015 to 0.30 inch before measuring. This increase in potential is also caused by immersion for a few minutes in dilute acids or by cathodic polarization. The measurements, the results of which are shown in the curves in Figs. 14 and 15, were made by the ordinary compensation method. The specimens were in all cases sandpapered or turned down after alkali treatment and paraffined for about one

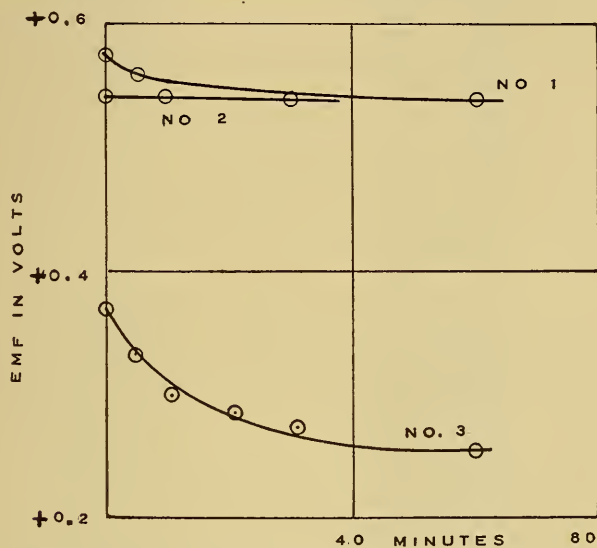


FIG. 15. EMF OF IRON TO NaOH BEFORE AND AFTER TREATMENT WITH 50 PER CENT NaOH AND WITH DILUTE H_2SO_4 (DR. HIRSCHKIND)

Curve 1....Emf after 10 minutes immersion in dilute H_2SO_4

Curve 2....Emf after one week treatment with 50 per cent NaOH at $110^\circ C$

Curve 3....Emf of original untreated material

inch below and above the surface of the solution while their potential was being measured. The curves, Fig. 15, are taken from Dr. Hirschkind's data; they show the potential in 1/10 normal sodium hydroxide of steel against Hg/HgCl. 1/10 normal KCl as a function of the time. Curve No. 3 gives the potential for the untreated steel; No. 2, for the same after a treatment of one week in 50 per cent sodium hydroxide at 100 degrees C; and No. 1, the potential after about ten minutes immersion in dilute sulphuric acid. It is to be noted that cathodic polarization gives the same effect as the sulphuric acid. The curves

in Fig. 14 also show this effect, the measurements being in 1/10 normal sodium hydroxide against Hg/Hg₀. 1/10 normal sodium hydroxide. No. 2 shows the potential of untreated material; and No. 1, the same after treatment in 13.6 N sodium hydroxide at 180 degrees for ten days. The potential is seen to be higher for all cases after treatment of any sort that evolves nascent hydrogen on the steel than for the original material. This fact would seem to point unmistakably to a molecular change in the iron.

This high potential seems to disappear in many cases after a lapse of time and after heating to from 100 to 200 degrees in air, but no relation was established between the presence of this potential and the existence of brittleness; the potential seemed to be increased during the treatment long before the brittleness manifested itself. It is, however, likely that there could be found such a parallelism, if the methods for testing the mechanical properties were as delicate as the potential measuring methods.

16. *Prevention of the Embrittling Effect.*—Another indication consistent with the hydrogen theory was found in studying means to prevent the deteriorative effect of sodium hydroxide. Of such means the first and most natural which suggests itself is the removal of the active agent.

If hydrogen is the cause of this brittleness, it should be possible to remove it by adding an oxidizing agent to the solution which would depolarize the iron electrode, as, for example, sodium dichromate. Preliminary measurements of the potential of iron to concentrated solutions of sodium hydroxide showed that at all temperatures readily accessible to such measurements (20 to 110 degrees), this potential was very considerably lowered by addition of the dichromate in equivalent proportions; that is, for one mol. of sodium hydroxide there was used 1/6 mol. of the dichromate. Curves showing this are given in Fig. 17; the measurements were made at 80 degrees C. Not only is Curve No. 1, which gives the potential of iron in 13.6N sodium hydroxide, higher (more electropositive) than No. 2 with the dichromate, but it also slopes upward, while No. 2 slopes the other way. Since the hydrogen potential in an alkaline solution of this concentration is calculated to be about $+0.930$, it would be impossible for it to be generated in the solution in which the $\text{Na}_2\text{Cr}_2\text{O}_7$ is present. Similarly, judging from these data, the corrosion in the dichromate solutions should be less noticeable than that in the sodium hydroxide

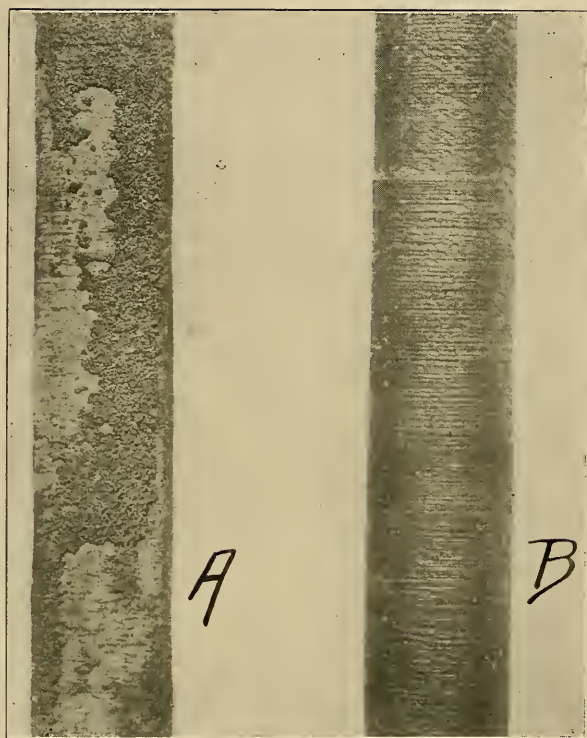


FIG. 16. THE CORROSION OF $\frac{1}{4}$ -INCH STEEL WIRE

A—After 1 week in 13.6 N NaOH at 280° C

B—After 1 week in $\left\{ \begin{array}{l} 13.6 \text{ N NaOH} \\ 13.6 \text{ N Na}_2\text{Cr}_2\text{O}_7 \end{array} \right\}$

solutions alone. In order to settle these points parallel physical tests were run in which a solution with equivalent amounts (13N) of both the alkali and the dichromate was used. The results of these tests are also presented in Tables 8, 9, and 10.

It was first noticed in opening the pipes after the treatment that although, as has already been mentioned, H_2 was present in those pipes which contained sodium hydroxide alone, only a few of those containing the dichromate had generated hydrogen during the treatment. This fact is shown in the tables. The corrosion in all cases was much less, being almost negligible in the case of less than a week's treatment at 180 degrees, and was, moreover, of an entirely different character. This is shown in the photograph Fig. 16. With pure sodium hydroxide the surface was rough and covered with a porous, loose deposit of black Fe_3O_4 , or a hydrated form of it, whereas the material deposited at the higher temperatures 180 to 280 degrees

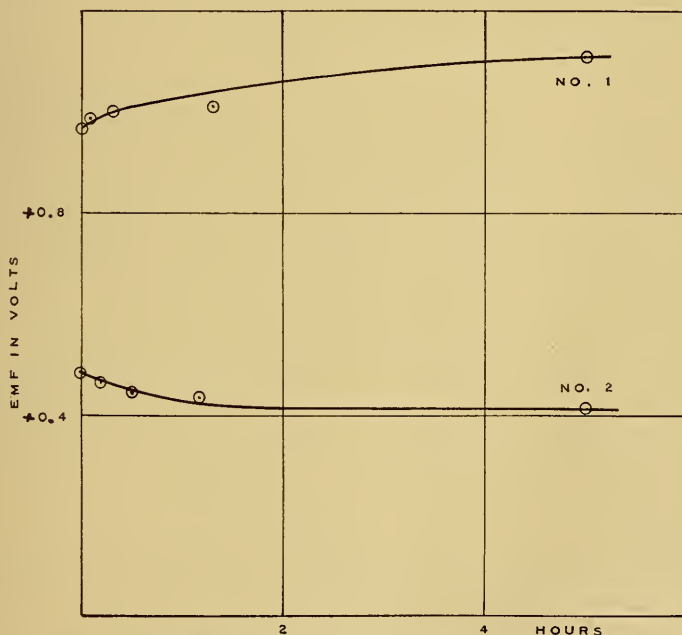


FIG. 17. EFFECT OF ADDITION OF SODIUM DICHROMATE UPON THE EMF OF IRON IN NaOH SOLUTION. TEMPERATURE 80 DEGREES C

Curve 1....Emf of iron to 13.6 N NaOH solution

Curve 2....Emf of iron to solution which is 13.6 N to both NaOH and $Na_2Cr_2O_7$

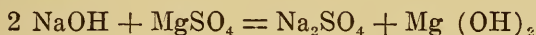
was of a beautiful black crystalline sheen. With the addition of $\text{Na}_2\text{Cr}_2\text{O}_7$ the surface was smooth and covered with a thin, dense, tightly adherent coating of brown Fe_2O_3 . In the latter case the marks of the lathe tool were generally still visible, as can be seen in the photograph.

The toughness, as indicated by the repeated bending test, apparently suffers no deterioration under the action of sodium hydroxide and sodium dichromate in the periods of treatment employed.

APPENDIX A

BOILER WATER TREATMENT

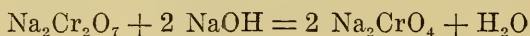
Magnesium Sulphate.—The simplest method for counteracting the excessive alkalinity of the water used at the University of Illinois power plant would seem to be by the addition of a salt having properties which would cause it to react with the alkali and yield a harmless product. Magnesium sulphate ($\text{MgSO}_4 + 7\text{H}_2\text{O}$) is such a salt, the reaction being:



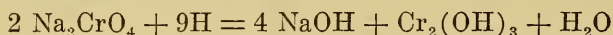
The resulting sodium sulphate remains in solution. The magnesium hydroxide, if the water is free from ammonia compounds, is insoluble and forms a precipitate or sludge. One pound of magnesium sulphate per 1,000 gallons of water will react with an equivalent of 3 grains per gallon of sodium carbonate. The water as used averages about 4.5 to 5 grains per gallon of sodium carbonate, requiring a theoretical treatment of about 1.5 pounds of MgSO_4 per 1,000 gallons.

Early in 1914 the use of magnesium sulphate was begun with amounts varying from $\frac{1}{2}$ to 2 pounds per thousand gallons. It was found practicable to reduce the sodium hydroxide alkalinity to zero. Facilities were lacking, however, to accomplish an exact treatment of the raw make-up water owing to the large quantity and varying amount of condensate returned from the heating system. Treatment was, therefore, very irregular; and at times when the treatment was discontinued, the sodium hydroxide concentration would occasionally reach 75 grains per gallon in the boilers. About January, 1916, separate tanks for the make-up water were made available and in these the treatment could be carried out in a way which would be independent of the return water. Much more concordant results were obtained and until March 1 the treatment was confined to the use of one pound of the sulphate to 1,000 gallons of water. The average alkalinity of the water in the boilers was less than 10 grains per gallon. Earlier experience with two pounds per thousand gallons had demonstrated that it was feasible to maintain the hydroxide alkalinity at practically zero when using that amount of sulphate.

Sodium Dichromate.—Another chemical which has been suggested in this connection is sodium dichromate. The reaction involved would be:



It is possible that this salt may serve a dual purpose, as indicated in Dr. Merica's discussion, by taking up the nascent hydrogen which might otherwise go into the iron, thus:



One feature was in evidence as a result of the use of this chemical. Considerable hydrolization took place within the boiler, probably promoted by the presence of the organic matter, so that some chromium was precipitated in the form of hydroxide $\text{Cr}_2(\text{OH})_3$, and this material mixed with other solids had a tendency to fuse or bake as an incrustation along the highly heated surfaces of the flues. While the use of this material was of advantage in indicating the location of leakage conditions in the boiler, its use was not continued long enough to prove its worth as a neutralizer of the hydrogen that might be formed.

Sulphuric Acid.—About March 1, 1916, sulphuric acid was used in an amount slightly more than the equivalent of 1 pound of magnesium sulphate per 1,000 gallons of water. At first the amount of acid was 0.39 pounds per 1,000 gallons. This was shortly increased to 0.54 pounds per thousand.

By reference to the composition of the water it will be seen that this amount of acid is sufficient to neutralize about 75 per cent of the sodium carbonate alkalinity, and that for the total alkalinity, including the magnesium and the calcium carbonates, the entire amount of acid required for neutralization would be about 2.5 pounds per 1,000 gallons, which leaves an ample margin of safety between the free acid employed and the total alkalinity of the water. This treatment was maintained steadily for 8 months. A quite uniform degree of sodium hydroxide alkalinity has resulted of approximately 9 per cent of the total solids present within the boilers. This has been accompanied by a sodium carbonate alkalinity of about 11 per cent of the total solids. This ratio of NaOH to Na_2CO_3 alkalinity is constant and evidently represents an equilibrium between the vapor and liquid phases of the constituents. A small amount of exudation with

accompanying incrustation has been collected from one of the new drums. An analysis of this material shows:

Sodium Carbonate, Na_2CO_3	= 15.3	per cent
Sodium Hydroxide, NaOH	= 1.3	“ “
Sodium Sulphate	= 72.7	“ “
Water	= 5.5	“ “

Assuming that the sodium carbonate before exposure to the air came from the boiler in the fairly constant ratio of forty-five parts of NaOH to fifty-five parts of Na_2CO_3 , we see that the active constituent, NaOH , is probably present at the point of exudation or contact with the boiler plate to the extent of about 8 per cent of the total solids. Whether this amount is low enough to preclude the possibility of hydrogen generation and consequent embrittling of the plate must remain for further inspection and experiments to determine. It may be said, however, that the new drums at the present time, after ten months of service with water that has been corrected for alkalinity, show no signs of trouble. The sudden rise in the price of magnesium sulphate and sodium chromate was the chief argument in deciding upon the use of the free acid. It offers an advantage also in that it adds no further solids to the water, and, from the viewpoint of the chemist, it is more logical to add a free acid which will form magnesium sulphate within the boiler than to add the more expensive magnesium sulphate already prepared. It may not be wise, however, to recommend the use of the free acid except under conditions where absolute control and reliable supervision are assured.

APPENDIX B

BIBLIOGRAPHY

Graham,* 1866, discovered that hydrogen penetrates red hot iron and when the iron cools down it retains a part of this hydrogen.

Cailletet† and Johnson‡ showed that iron at ordinary temperature absorbs electrolytically generated hydrogen, while gaseous hydrogen has no effect at all. Johnson's investigation is very interesting. He used soft iron wire which became brittle as soon as

*Proc. Roy. Soc., vol. 17, p. 219, 1869.

†Compt. rend., vol. 80, p. 319, 1875.

‡Proc. Roy. Soc., vol. 23, p. 186, 1875.

electrolytic hydrogen was generated on it. If the wire was dipped in acid, it became brittle immediately.

Bellati and Lussana * used a barometer, closed with an iron plate, which was at the same time a cathode in an electrolytic cell. The generated hydrogen was diffused through the iron plate, as indicated by the falling of the barometer. The results of Bellati and Lussana were also verified by Shields.†

Heyn ‡ carried out a series of investigations on the effect of hydrogen on mild open hearth steel and found: first, that if a piece of steel was heated in contact with hydrogen to a temperature of from 730 to 1,000 degrees and was suddenly quenched in water, the metal became decidedly more brittle than if heated in the open air before quenching, and that slow cooling in hydrogen or heating to less than 330 degrees before quenching, had no effect which could be detected by a bending test; secondly, the hydrogen penetrated gradually from the surface inwards, but the effect could be neutralized by heating the specimens afterwards in air or in contact with nitrogen; thirdly, if the pieces rendered brittle by heating in hydrogen and subsequent quenching were exposed to the atmosphere for a long time, the toughness was restored wholly or in a great part. It was not possible to detect the least difference between the micro-structure of specimens heated in air and quenched.

E. Heyn gives further details of his experiments on the influence of hydrogen on iron. The brittleness of the iron, due to the hydrogen, can be very readily overcome. If the iron is heated very slightly or boiled in water or in oil, this brittleness is either wholly or in part eliminated. After the metal had been exposed to the air for fourteen days without any extra heating the brittleness of the open hearth iron completely disappeared, but in the case of the wire, also extremely low in carbon, the metal was as brittle at the end of fourteen days as initially. In 251 days the hardness and brittleness were nearly eliminated.

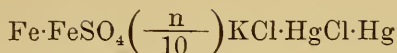
A very important contribution to the subject has been given by T. W. Richards and G. E. Behr § concerning the electromotive force of iron under different conditions and the influence of occluded hydrogen. As the first of many investigators, they determined the potential of an iron electrode in a normal solution of ferrous ions in the cell:

*Z. physik. Chem., vol. 7, p. 229, 1891.

†Chemical News, vol. 65, p. 195, 1892.

‡Stahl und Eisen, vol. 20, p. 837, 1900.

§Z. physik. Chem., vol. 58, p. 301, 1907.



They found as the electromotive force 0.77 volt against the decinormal electrode for compact iron; for porous iron and iron sponge they found it to be 0.03 volt higher, namely, 0.8 volt. This is probably due to the larger surface in the latter case. It was also found that the enormous pressure of 350,000 kg. to the square centimeter caused no variation of the potential; that sudden cooling from a very high temperature had no influence; and that iron powder reduced at low temperature can take up considerable quantities of hydrogen without any change of the potential. Contrary to E. Heyn's statement, previously described, they found that iron heated in hydrogen above 730 degrees and rapidly cooled does not take up any hydrogen, but if the heated iron is rapidly cooled in water, active hydrogen is taken up which considerably increases the potential. They stated further that in this case it would not make any difference if the iron was heated in a hydrogen or nitrogen atmosphere. The hydrogen, which is taken up by the metal in sudden cooling in water, is driven out in the ferrous sulphate solution and the potential goes down to the normal value. The hydrogen in this case seems to be in any active form and equal to the nascent hydrogen which is taken up by the iron (Johnson and Cailletet) whether the hydrogen is generated chemically or electrolytically. E. Heyn in his reply objects to Richards' and Behr's statement that no active hydrogen is taken up by heating in a hydrogen atmosphere. This divergence is explained by the fact that Richards and Behr experimented with spongy iron and Heyn with compact iron.

Concerning the diffusion, occlusion, and solubility of hydrogen in iron we owe two careful studies to A. Sieverts.* He found that at constant temperature the solubility of hydrogen in solid and liquid metal is proportional to the square root of the gas pressure. At constant gas pressure the solubility of hydrogen is increased with the temperature. At the melting point the increase of the solubility is discontinuous. The transition of X into B iron and of B into Z iron cannot be recognized on the absorption curve.

If solid iron is slowly cooled down in hydrogen the dissolved hydrogen is nearly completely liberated. But if it is cooled down

*Z. physik. Chem., vol. 60, p. 153, 1907.

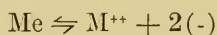
rapidly, the hydrogen stays in the iron. Sieverts found that in iron heated to 800 degrees and slowly cooled down 0.16 per cent H_2 is liberated. Heyn found 0.16 per cent H_2 in a piece of iron which was treated in a hydrogen atmosphere at from 750 to 800 degrees and rapidly cooled down, which agrees with Sieverts results.

Charpy and Bonnerot,* two French investigators, studied the reactions accompanied by the osmosis of hydrogen through iron. They found no specifically new facts, but stated that molecular hydrogen penetrating through iron has no effect on its structure and strength.

Concerning the phenomenon which is called passivity of iron there is much literature, but only the most recent investigations are here noted.

The metals can be divided into two groups: The first that in which the potential difference between metal and electrolyte is purely a function of the concentration and temperature of the electrolyte; and the second (Fe, Ni, Cr) that in which the potential depends also on the state of the metal itself. We will not discuss all the numerous and different theories of passivity, but only the most recent one, which, it is true, is not as yet generally accepted, but which seems to give promise of proving correct.

A passive metal can be defined without respect to any theory as a metal in that state in which it does not send ions into the solution. That is to say, the reaction



takes place with an extremely slow velocity. Foerster † first brought up the question as to whether the active state is due to a catalyzer. All the metals which exist in an active and passive state like iron, nickel, cobalt, or chrome take up hydrogen very easily. Electrolytic iron contains much hydrogen taken up by the metals which effects the active state as a catalyzer in the reaction described.

E. Grave ‡ claims as the result of his experiments that it is only the hydrogen ion which effects the active state in metals.

J. H. Andrew § describes a series of experiments wherein he was able to demonstrate the brittleness of strips of steel after submerging

*Compt. r., vol. 156, p. 394, 1913.

†Abh. der deutschen Bunsenges., vol. 2, p. 25, 1909.

‡Z. physik. Chem., vol. 77, p. 513, 1911.

§Transactions Faraday Society, p. 316, March, 1914.

them in concentrated sodium hydroxide at 100 degrees for from one to seven weeks. The action of the caustic soda was accompanied by the evolution of hydrogen. The embrittling effect was more noticeable in the initial stages of the experiment and increased during the first few weeks of the treatment. This brittleness ultimately disappeared and malleability was again restored. These characteristics are especially interesting in the light of Dr. Merica's experiments with sealed solutions of caustic.

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ISSUED WEEKLY

Vol. XIV

JANUARY 29, 1917

No. 22

Entered as second-class matter Dec. 11, 1912, at the Post Office at Urbana, Ill., under the Act of Aug. 24, 1912.]

MAGNETIC AND OTHER PROPERTIES OF IRON-ALUMINUM ALLOYS MELTED IN VACUO

BY

TRYGVE D. YENSEN

AND

WALTER A. GATWARD



BULLETIN No. 95

ENGINEERING EXPERIMENT STATION

PUBLISHED BY THE UNIVERSITY OF ILLINOIS, URBANA

PRICE: SEVENTY CENTS

EUROPEAN AGENT

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JANUARY, 1917

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OF IRON-ALUMINUM ALLOYS
MELTED IN VACUO

BY

TRYGVE D. YENSEN

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MAGNETIC AND OTHER PROPERTIES OF IRON-ALUMINUM ALLOYS MELTED IN VACUO

I. INTRODUCTION

IT is the purpose of this bulletin to record the results of experiments to determine the magnetic and allied properties of iron-aluminum alloys melted in vacuo. Previous investigations to determine these properties of pure iron,* iron-boron alloys,† and iron-silicon alloys,‡ melted in vacuo, have been reported in earlier bulletins. Because of the similarity of many properties of aluminum and silicon, the same remarkable results were expected from the iron-aluminum as from the iron-silicon alloys. To some extent these have appeared. The results obtained, although not covering every phase of the subject, include the magnetic, electrical, and mechanical properties. Chemical analysis and photomicrographs are also presented for a number of the alloys.

The chemical analysis was in charge of Mr. J. M. Lindgren of the Chemistry Department; the methods used for the determination of aluminum are described by Mr. Lindgren in the Appendix. The photomicrographs were prepared by Mr. F. E. Rowland, also of the Chemistry Department. Mr. H. R. Fritz, Research Fellow in the Engineering Experiment Station, has rendered valuable service throughout the year as general assistant in connection with the investigation. Besides acknowledging their indebtedness to those persons who have been directly connected with the work, the authors also wish to express their appreciation to many others who have been of service to them. Among these it is particularly desired to mention Professor E. B. Paine, acting head of the Electrical Engineering Department.

Aluminum was first used in the industries in connection with copper to form aluminum bronzes, and it was this application of it that stimulated investigators to develop methods of manufacture capable of producing it at a cost which would warrant its use in large quantities. It is well known today that aluminum occurs in nature

*"Magnetic and Other Properties of Electrolytic Iron Melted in Vacuo." Univ. of Ill. Eng. Exp. Sta., Bul. 72, 1914. Trans. Am. Inst. Elect. Engrs., vol. 33, I, p. 451.

†"The Effect of Boron Upon the Magnetic and Other Properties of Electrolytic Iron Melted in Vacuo." Univ. of Ill. Eng. Exp. Sta., Bul. 77, 1915.

‡"Magnetic and Other Properties of Iron-Silicon Alloys Melted in Vacuo." Univ. of Ill. Eng. Exp. Sta., Bul. 83, 1915. Proc. Am. Inst. Elect. Engrs., vol. 34, p. 2455. Oct., 1915. Proc. Am. Inst. Mining Engrs., p. 482, Feb., 1916.

only in the form of oxides and that it is one of the stablest oxides to be found, so stable that the only way of reducing it to its metallic state is by means of electrolysis. This fact early suggested its use for the purpose of reducing other oxides, and it was in the capacity of a reducer, deoxidizer, or degasifier, that it very early became the friend of the iron and steel maker. For this purpose aluminum soon became a serious competitor of silicon, in spite of the high cost of aluminum in its early days. Many were the wonderful qualities attributed to this new metal. A large number of these, however, existed only in the imagination of their advocates and were soon disproved.

Among the well-established properties of aluminum, the following are perhaps the most important: *

a. Aluminum when added to molten iron will reduce iron oxide to metallic iron. Furthermore, it will reduce CO or CO₂ gases to free carbon and thus degasify, quiet the bath, and prevent blowholes during solidification and cooling. If the iron is sufficiently liquid, the Al₂O₃ formed will pass to the surface; if it is not, the Al₂O₃ will become entangled in the iron, increase the viscosity, and cause the iron to be more or less unsound.

b. When added in sufficient quantities, 2 to 5 per cent, aluminum will cause carbon to be completely precipitated as graphite and thus transform, for example, white cast iron into gray iron. If 10 to 20 per cent aluminum is added, however, the carbon may all be in the combined form Fe₃C.

c. As the heat of combustion liberated by the oxidation of aluminum is much greater than that absorbed by the reduction of iron oxides, clearly illustrated by the use of thermite in welding iron, the addition of aluminum to oxidized iron or steel causes a rise in the temperature of the bath. This fact, and not any appreciable lowering of the melting point, is the cause of the higher fluidity noticed when a small percentage of aluminum is added to the bath.†

The metallographic investigation of iron-aluminum alloys was first systematically undertaken by the Alloys Research Committee

*Birmingham Eng. Jour., vol. 3, pp. 138-145. Transactions Am. Inst. Min. Engrs., vol. 18, pp. 102-122, 1889-90. Journal Iron and Steel Inst., vol. 37, I, pp. 112-133, 1890. Transactions Am. Inst. Min. Engrs., vol. 18, pp. 835-58, 1889-90. The Eng. and Min. Jour., vol. 50, p. 213, rev. vol. 42, p. 265. Iron & Steel Trades Jour., vol. 43, pp. 309, 399. Transactions Am. Inst. Min. Engrs., vol. 20, p. 233, 1891. Jour. Soc. of Chem. Ind., vol. 12, p. 239, 1893.

†Journal Iron and Steel Inst., vol. 38, II, pp. 161-230, 1890.

of the Institution of Mechanical Engineers,* and later, more extensively, by Gwyer.† The equilibrium diagram according to the latter is shown in Fig. 1. This diagram shows that iron and aluminum, when liquid, dissolve in each other in any proportion. Up to an

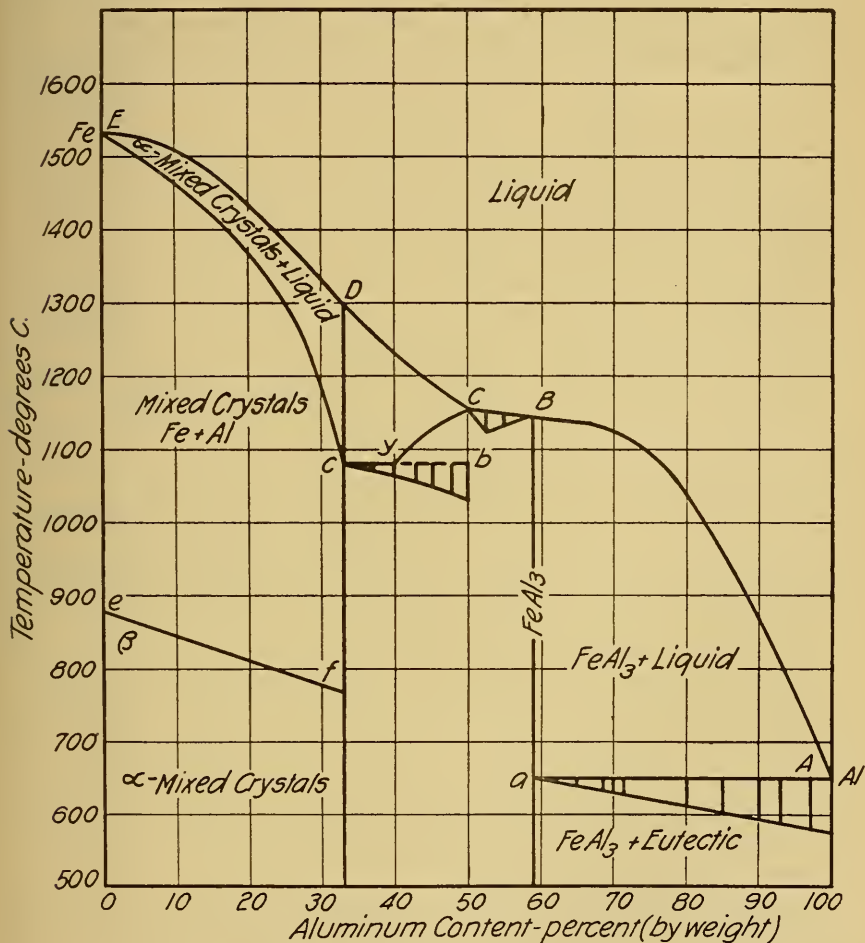


FIG. 1. EQUILIBRIUM DIAGRAM OF IRON-ALUMINUM ALLOYS ACCORDING TO GWYER

aluminum content of 50 per cent mixed crystals are at first formed upon solidification, but as the solid solution becomes saturated with

*Rpt. Alloys Research Committee, Proc. Inst. Mech. Engrs., No. 2, pp. 238-297, 1895.

†Z. Anorg. Chem., vol. 88, pp. 113-53, 1908.

an aluminum content of 34 per cent, any excess above this amount will be precipitated as solidification proceeds. Alloys containing less than 34 per cent aluminum will suffer no precipitation of any constituent during cooling and, therefore, at ordinary temperatures will consist of mixed crystals of iron and aluminum. The lowering of the melting point of iron due to small percentages of aluminum is very slight. Even for a 10 per cent aluminum content the melting point is lowered only about 10 degrees. This shows conclusively that the increased fluidity of iron, previously referred to, which is obtained by adding less than 0.1 per cent aluminum, can not be due to the lowering of the melting point. With an aluminum content of 59 per cent, the compound FeAl_3 crystallizes out during solidification and leaves behind liquid aluminum, which freezes at 653 degrees. The theory suggested by previous writers, that the liberation of heat noticed upon adding a small amount of aluminum to molten iron might be due to the formation of compounds of iron and aluminum,* is, therefore, at variance with the evidence just presented. The only explanation of the described phenomenon which has stood the test of investigation is that the heat liberated is due to the formation of Al_2O_3 .

From these investigations it appears that within the region studied in the present research, namely 0 to 10 per cent aluminum, the alloys all consist of mixed crystals of iron and aluminum, just as the iron-silicon alloys described in Bulletin 83 all consisted of mixed crystals of iron and silicon. The photomicrographs of Gwyer, Guillet, and Hadfield confirm the deductions just stated.

The mechanical properties of iron-aluminum alloys have been studied by Hadfield,† Styffe,‡ and Guillet,§ who all agree that aluminum in quantities up to 5 per cent has very little influence upon the mechanical properties of iron. When more than 5 per cent is used, however, the alloys are slightly stronger in tension, but become brittle. Hadfield, whose alloys contained about 0.2 per cent carbon, found the limit of forgeability between 5.6 and 9.14 per cent aluminum content, while Guillet found the limit between 7.18 and 9.25 per cent for alloys containing 0.8 per cent carbon. The results by these three investigators are shown graphically in Fig. 6.

The effect of aluminum on the magnetic properties of iron was

*Journal Iron and Steel Inst., vol. 38, II, p. 170, 1890.

†Journal Iron and Steel Inst., vol. 38, II, pp. 161-230, 1890.

‡Oestereich Z., für das Berg—Hütten—und Salinenwesen, vol. 41, p. 536, 1893.

§Rev. de Metal. Memoirs, vol. 2, pp. 312-327, 1905. Journal Iron and Steel Inst., vol. 70, II, p. 16, 1906.

studied to some extent in the early nineties * after the beneficial effect of this element as a purifier of iron and steel had been definitely established. It remained for Hadfield,† however, to make the first iron-aluminum alloys which had better magnetic qualities than those of the purest iron obtainable at the time. His 2.25 and 5.5 per cent aluminum alloys as tested by Barrett showed a higher permeability and a lower hysteresis loss than did the purest Swedish charcoal iron (see Table 1).

TABLE 1
MAGNETIC AND ELECTRICAL PROPERTIES OF HADFIELD'S IRON-ALUMINUM ALLOYS

MARK	Al	Max. Permea- bility	Permea- bility for H=8	Max. Induction for H=45	Reten- tivity in Terms of B	Coercive Force in Terms of H	Hysteresis loss Ergs per c. c. per cycle		Specific Electrical Resist- ance Microhms
							for H=45	for B=9000	
S.C.I.	.60	4 000	1 700		10 800	1.10	abt. 8 500	2 334	10.2
B	.00		1 625	16 800	9 770	1.66	10 760		10.9
1167 D	.75		1 500	16 000	10 500	1.80	11 000		22.0
1167 H	2.25	6 000	1 700	16 900	10 500	1.00	8 000	1 443	39.0
1167 I	5.50		1 200	13 000	4 150	1.00	6 500		70.0

From the results shown in the table and those previously obtained with silicon,‡ aluminum and silicon appear to have very similar effects upon the magnetic properties of iron.

Dillner and Engstrom § have investigated the effect of aluminum and silicon on the magnetic properties of sheet iron as well as of castings. They found that silicon reduced the permeability and increased the hysteresis loss in sheets, but not in castings, and that aluminum had the opposite effect. Furthermore, it was found that a combination of silicon and aluminum gave the best results for sheets.

The main objection to employing aluminum formerly was its high cost. While silicon, in the form of ferrosilicon, could be obtained at a cost of 12 cents per pound of the element, the cost of aluminum was nearly \$2 per pound. On this account silicon was employed in making steel for magnetic purposes, and, having given universal satisfaction both with regard to the hysteresis loss and to aging, it has retained its place. In spite of the reduced cost of aluminum, there

*Transactions Am. Inst. Elect. Engrs., vol. 9, pp. 250-262 (and Discussion). The Electrician, vol. 29, p. 475, 1893.

†Scientific Trans. Royal Dublin Soc., vol. 7, ser. 2, pp. 67-126, Jan., 1900. Journal Inst. Elect. Engrs., vol. 31, pp. 674-729, 1902.

‡Scientific Trans. Royal Dublin Soc., vol. 7, ser. 2, pp. 27-126, Jan., 1900.

§Journal Iron and Steel Inst., vol. 67, I, pp. 474-480, 1905.

has appeared to be no decided advantage in changing from silicon steel to aluminum steel. Investigators have turned their attention toward improving and perfecting silicon steel while comparatively little investigational work has been recorded regarding iron-aluminum alloys, particularly regarding their magnetic properties. It is hoped, therefore, that the present investigation will prove to be of interest not only on account of the vacuum method employed, but also because it is one of the few systematic investigations on the subject.

II. MATERIAL, APPARATUS, AND METHODS *

The iron used as the basis of the investigation was doubly refined electrolytic iron containing 0.01 per cent carbon or less and about 0.01 per cent silicon. It was cleaned with HCl, distilled water, and alcohol, and then dried by means of ether in vacuo.

The aluminum used as the alloying material analyzed as follows:

Si	0.20	per cent
Fe	0.17	“ “
C	nil	
Al (by diff.).....	99.63	“ “

The melting was done in magnesia crucibles in an Arsem type vacuum furnace capable of melting 600 grams of iron; the pressure was less than 1 mm. of mercury. Attempts were first made to melt the iron and the aluminum together, but this method had to be given up for two reasons. First, the melting point of aluminum is so low compared with that of iron, that a considerable portion of aluminum evaporated before the iron was in a condition to combine with it. Furthermore, the aluminum vapor interfered with the operation of the furnace in such way that the voltage could not be maintained at a sufficiently high value to melt the iron completely. The result was that the Al_2O_3 formed, instead of rising to the surface, became entangled in the iron, often completely enclosing pieces of iron and resulting in imperfect ingots. Second, as will be shown, on account of the power of aluminum at high temperatures to reduce CO gas, this method of melting was certain to introduce more or less

*For further information see "Magnetic and Other Properties of Electrolytic Iron Melted in Vacuo." Univ. of Ill. Eng. Exp. Sta., Bul. 72, 1914; and "Magnetic and Other Properties of Iron-Silicon Alloys Melted in Vacuo." Univ. of Ill. Eng. Exp. Sta., Bul. 83, 1915.

carbon into the alloys. It became necessary, therefore, first to melt the iron and then, towards the end of the operation, to drop in the aluminum. This was accomplished without opening the furnace by suspending the aluminum (in the form of a wire or rod) from a very fine wire extended between insulated binding posts which passed through the cover of the furnace. In order to drop the aluminum, the fine wire was fused by connecting the binding posts for a moment to the terminals of the furnace. This method made it necessary to use a cover with a hole in it over the crucible, together with a funnel to direct the rod into the crucible.

It has already been mentioned as a well established fact that aluminum at high temperatures, when added to molten iron for instance, will reduce CO gas and thus liberate free carbon. That it will do so even at lower temperatures has been shown by Stead,* and in the present investigation this has been further confirmed.

It is evident that whatever gases remain in the vacuum furnace must consist largely of carbon monoxide. During the first trial experiments with the aluminum suspended from the top of the furnace, it was noticed, upon examining the aluminum after the operation, that the lower end of the wire was coated with a grayish substance. This substance was evidently the mixture of carbon and Al_2O_3 mentioned by Stead. The lower the wire reached, and consequently, the hotter it became, the larger the portion of it which became coated. In order to prevent the wire from becoming hot, it was made as short as possible and, during the later part of the investigation, was protected from direct radiation by means of an iron tube extending down into the furnace. But even with these precautions the tip of the wire or rod was always slightly coated, particularly when more than 5 per cent aluminum was used. Moreover, when the aluminum was dropped in, it obviously became hot and may have had time in a few cases to reduce some CO. In these cases a small amount of carbon was invariably introduced into the alloy, generally not exceeding 0.02 per cent, but in some cases, especially during the early part of the investigation, amounting to 0.10 per cent or even more. One alloy analyzed as high as 0.65 per cent carbon.

It is quite evident from this discussion that in the type of furnace used it would be out of the question to mix the iron and the aluminum, melt them together, and expect to obtain an alloy with

*Journal Iron and Steel Inst., vol. 38, II, p. 190, 1890.

a low carbon content. At first this method was used without bad effects for very low percentages of aluminum when the iron oxide present evidently was sufficient to oxidize both the aluminum and the carbon, but the method was abandoned for aluminum contents above 0.5 per cent, for the reasons already stated.

The alloys known or suspected to contain more than 0.015 per cent carbon for low aluminum alloys and more than 0.025 per cent carbon for high aluminum alloys have been tabulated separately in order to show the effect of carbon upon the various properties. It appears that the methods employed were not ideal, and that small amounts of carbon may have been introduced with the aluminum. It is possible, therefore, that better results, magnetically, might have been secured had a furnace been employed, having, for instance, a tungsten heating element.

The ingots were allowed to cool while in vacuo, and then were forged into rods and machined into test pieces.

The magnetic testing was done by means of the Burrows compensated double bar and yoke method. It has been previously shown* that the errors made by using this method for very high permeability iron may be much greater than theoretical considerations would indicate, but no corrections have been made in the present case, it being desired that the results given in this bulletin be comparable with those given in the previous bulletin on iron-silicon alloys. Furthermore, the question of corrections is a complicated one and has by no means been definitely settled, for the amount of compensating current needed to equalize the flux along the rod depends largely upon the uniformity of the rods used. This question is further discussed in the Appendix. The Appendix contains also additional data obtained from ring specimens, showing that for permeabilities not exceeding 20,000, the results obtained with the Burrows method may be relied upon as being very close to the correct values.†

The tests for electrical resistance were made by means of a Kelvin double bridge, using standardized iron rods for comparison.‡ For the mechanical testing an Olsen 10,000 pound testing machine was employed, and the usual characteristics were obtained. The rods

*"Magnetic and Other Properties of Iron-Silicon Alloys Melted in Vacuo." Univ. of Ill. Eng. Exp. Sta., Bul. 83, pp. 24-27, 1915.

†"Magnetic and Other Properties of Iron-Silicon Alloys Melted in Vacuo." Univ. of Ill. Eng. Exp. Sta., Bul. 83, Appendix, 1915.

‡The standardization was done by the Leeds and Northrup Co.

were tested magnetically and electrically after the following heat treatments:

- a. As forged
- b. Annealed at 900 degrees C. Cooled at the rate of 30 degrees per hour
- c. Annealed at 1,100 degrees C. Cooled at the rate of 30 degrees per hour

Mechanical tests were made on unannealed specimens and on specimens heated to 1,000 degrees C. and then cooled at the rate of 30 degrees C. per hour. The heat treatment was applied in vacuo in order to prevent oxidation, a vacuum of 0.2 to 0.1 mm. being maintained during the treatment. The specimens were packed in powdered magnesia in an electroquartz tube.

III. CHEMICAL ANALYSIS

The marked effect of aluminum and silicon upon the electrical resistance of iron is well known. One per cent of either element added to iron will more than double its electrical resistance, and the increase is nearly proportional to the amount added. Furthermore, the resistance is not appreciably affected by mechanical or thermal treatments. These facts furnish an excellent opportunity for establishing the aluminum content of iron which contains only small amounts of impurities. This opportunity has been used to advantage in the present case. A few alloys were selected the aluminum content of which, as determined approximately by the aluminum added to the iron, covered the range studied. After the magnetic and electrical tests on these alloys were completed, the test rods were placed in a lathe, the surface layer was removed, and fine shavings were collected from along the rods in sufficient quantities for duplicate chemical analysis.

The results of this analysis * are given in Table 2, Column 3; from these figures and those for the electrical resistance, given in Column 5, the curve in Fig. 2 showing the relation between the resistance and the aluminum content was established. The aluminum content of all the other alloys for which electrical resistance measurements could be made, was then determined by means of the curve in Fig. 2. The values thus obtained have been checked by further analysis and found to be within the limits of accuracy desired.

*The method used is described by Mr. J. M. Lindgren in the Appendix.

It was stated in the previous chapter that, due to the conditions of melting and to the reduction of CO by aluminum, a large number of the alloys contained carbon. Therefore, a careful weeding of the alloys was made.

TABLE 2
LIST OF ALLOYS AND THEIR CHEMICAL ANALYSIS

No.	Aluminum Content		Carbon Content Per Cent by Chem. Analysis	Specific Elec. Resistance Annealed at 1100° C. Mi- crohms	Alumi- num Content from Res. Curve Per Cent	Remarks
	Per Cent As Charged	Per Cent by Chem. Analysis				
3 Al. 05	0.1	0		10.2	0.02	} Aluminum mixed with iron before melting. } Classed as contaminated.
06	0.2	0		10.4	0.05	
07	0.3	Trace		10.4	0.05	
08	0.4	Trace		9.9	0.00	
09	0.5	0.09	0.05	11.0	0.09	
10	0.6	0.11		12.0	0.18	
11	0.7	0.09		10.9	0.08	
12	1.0			14.4	0.40	
18		0.11		12.6	0.23	
19	0.0			9.7	0.00	
20	1.0		0.01	19.2	0.80	} Classed as contaminated.
21	2.0		0.01	30.2	1.80	
22			0.02	15.3	0.45	
23	3.0		0.01	39.7	2.67	
24	4.0	3.53	0.06	49.0	3.58	
25	5.0	4.83	0.16	57.7	4.52	
26		7.63	0.45			
27	7.6	8.10	0.09	81.4	7.95	
28		12.36	0.65			
29		13.05	0.38			
30	0.2			11.0	0.09	} Classed as contaminated. } Only small portion of added Al. } apparently dissolved in the iron. } Consequently carbon believed to be high.
31	0.4			11.1	0.09	
32	0.6			11.6	0.10	
33	0.8			14.1	0.37	
34	1.0		0.02	12.4	0.22	
35	1.5			15.0	0.44	
36	2.0	0.73		18.0	0.69	
37	2.5			19.0	0.78	
39	3.0			13.7	0.32	
40	3.4	3.31		43.9	3.08	
41	3.9			57.8	4.54	} Classed as contaminated.
42	4.4			51.2	3.82	
43	4.8			57.9	4.55	
45	5.7			63.5	5.20	
46	6.7	5.30	0.02	67.7	5.72	
47		8.60	0.13	84.3	8.75	
49	6.6			70.4	6.24	
50		1.55		29.4	1.70	
51	7.7		0.11	66.1	5.53	

The alloys obtained by mixing the aluminum with the iron before melting includes Nos. 3 Al 05-17. Of these, only Nos. Al 05-08 were retained as being uncontaminated; the ingots for Alloys Nos. 3 Al 13-17 were decidedly unsound. Those in the next series, 3 Al 18-29, inclusive, were prepared by dropping the aluminum into the molten iron. Most of the alloys of this series contain large amounts of carbon, for the aluminum was not protected from undue heating.

Furthermore, when a large piece of aluminum was dropped into the crucible, such agitation resulted that some of the molten iron undoubtedly came in contact with the small graphite funnel, which was used to guide the aluminum into the crucible. Finally, during the melting of the Alloys 3 Al 26, 28, and 29, the MgO crucible cracked

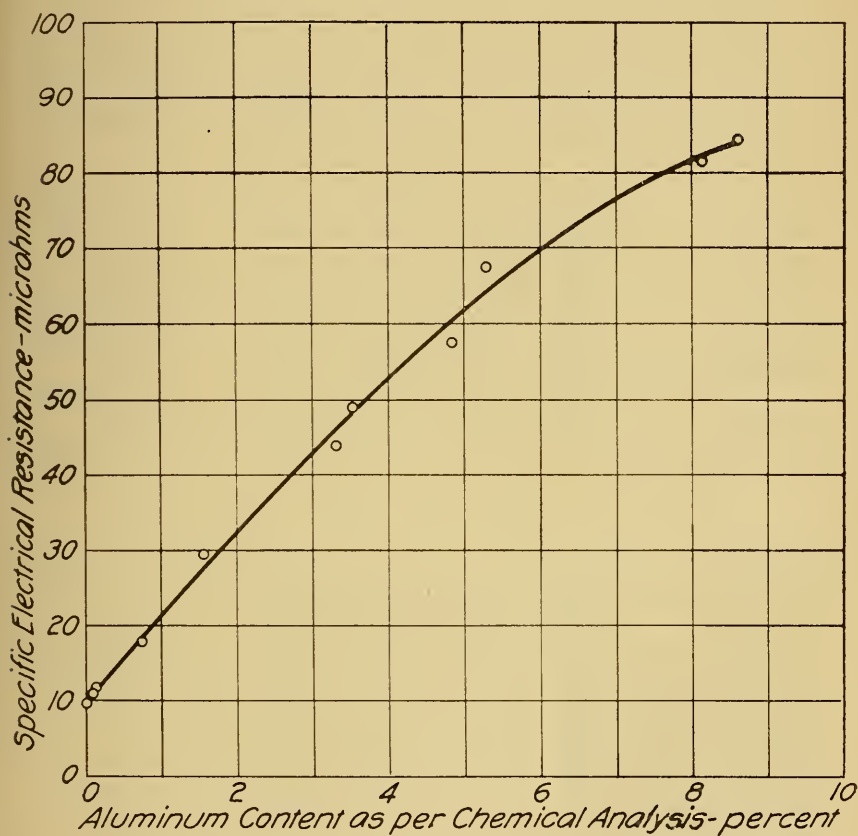


FIG. 2. ELECTRICAL RESISTANCE OF IRON-ALUMINUM ALLOYS MELTED IN VACUO. ANNEALED AT 1100 DEGREES C.

and the iron came in contact with the graphite container. In the last series, 3 Al 30-51, inclusive, an aluminum or magnesia funnel was used, and the aluminum was protected from undue heating by means of an iron tube. In this series, 3 Al 34-39, inclusive, were classed as contaminated because the aluminum content of the iron was, for

TABLE 3
RESULTS OF MECHANICAL TESTS. AS FORGED

Spec. No.	Alumi- num Content Per Cent	Carbon Con- tent Per Cent	Yield Point lbs. per sq. in.	Ult. Str. lbs. per sq. in.	Elongation Per Cent		Reduc- tion of Area Per Cent	Remarks
					Before Neck- ing	Ulti- mate		
3 Al. 19	0.00	...	50700	54700	4.0	26.0	84.3	
05	0.02	...	40200	46400	5.0	28.0	93.4	
31	0.09	...	43400	46200	3.0	28.0	88.4	
20	0.80	.01	86300	86500	...	25.0	86.9	Broke outside punch marks.
50	1.70	...	79100	83400	3.0	3.0	6.8	
21	1.80	.01	63600	64800	3.0	20.0	82.8	
23	2.67	.01	47700	59700	13.0	36.0	81.6	Broke outside punch marks.
40	3.08	...	68200	77500	4.0	21.0	76.4	
42	3.82	...	73400	77800	3.0	25.0	82.7	
41	4.54	...	57600	66300	8.0	28.0	82.6	
43	4.55	...	81800	84400	2.5	23.0	81.6	
45	5.20	...	80000	82800	1.0	20.0	84.3	
46	5.72	.02	78400	86700	5.0	26.0	82.6	
49	6.24	...	77700	86000	5.0	28.0	74.7	

CONTAMINATED ALLOYS

3 Al. 11	0.08	...	39200	46360	11.0	33.0	89.8	
34	0.22	.02	58600	64400	1.0	12.0	55.0	
39	0.32	...	84800	88700	3.0	22.0	79.6	
35	0.44	...	62000	72100	6.0	22.0	73.5	
36	0.69	...	77700	83300	8.0	25.0	82.5	
37	0.78	...	79700	84900	3.0	20.0	73.6	Broke outside punch marks.
24	3.58	.06	61400	75300	7.0	25.0	76.8	Broke outside punch marks.
25	4.52	.16	90800	96600	1.0	1.0	2.5	
51	5.53	.11	89100	96600	0.0	0.5	2.1	
26	7.63	.45	71900	71900	1.0	1.0	0.5	
27	7.95	.09	72800	84100	11.0	27.0	76.6	
47	8.75	.13	64900	73600	5.0	18.0	36.3	
28	1312.6	.65	85200	90500	1.5	1.5	7.5	Broke outside punch marks.
29	03.5	.38	96500	115000	0.0	0.5	2.6	

TABLE 4

RESULTS OF MECHANICAL TESTS. ANNEALED AT 1000 DEGREES C.

Spec. No.	Alumi- num Content Per Cent	Carbon Con- tent Per Cent	Yield Point lbs. per sq. in.	Ult. Str. lbs. per sq. in.	Elongation Per Cent		Reduc- tion of Area Per Cent	Remarks
					Before Neck- ing	Ulti- mate		
3 Al. 19	0.00		17600	34900	33	60	93.5	Flaw in Test piece.
08	0.00		14000	32100	29	48	65.7	
05	0.02		13900	34470	32	60	91.6	
30	0.09		13800	36400	27	50	84.3	
31	0.09		13700	33700	28	62	89.0	
32	0.10		14200	35900	31	57	91.6	
20	0.80	0.01	21700	40100	26	56	91.5	
21	1.80	0.01	23900	46000	27	61	90.7	
23	2.67	0.01	30100	49400	28	49	89.0	
40	3.08		31800	53400	25	51	85.3	
42	3.82		37300	55900	24	51	85.1	{ Broke at punch mark. { Data not plotted.
41	4.54		18100	43800	21	54	85.9	
43	4.55		41900	60200	22	49	83.8	
45	5.20		45000	61600	20	43	83.8	
46	5.72	0.02	50200	67400	18	40	79.7	
49	6.24		53400	69800	11	27	55.5	

CONTAMINATED ALLOYS

3 Al. 09	0.09	0.05	21000	37900	33	60	85.1	Broke outside punch mark.
10	0.18		18800	37100	24	53	79.8	
34	0.22	0.02	13500	37700	25	51	83.0	
39	0.32		50700	67500	14	32	79.2	
35	0.44		21000	46800	28	50	83.8	
36	0.69		34900	55300	24	49	83.0	
37	0.78		38300	56800	24	48	86.9	
24	3.58	0.06	39500	56900	27	54	76.8	
25	4.52	0.16	39200	60800	22	46	76.9	
51	5.53	0.11	79600	84800	1	1	2.2	
26	7.63	0.45	58400	63200	3	3	2.2	Broke outside punch mark.
27	7.95	0.09	26100	50300	20	45	89.5	
47	8.75	0.13	24900	50300	21	50	85.6	
28	12.36	0.65	26100	59000	4	4	6.1	
29	13.05	0.38	91500	91500	1	1	0.0	Broke outside punch mark.

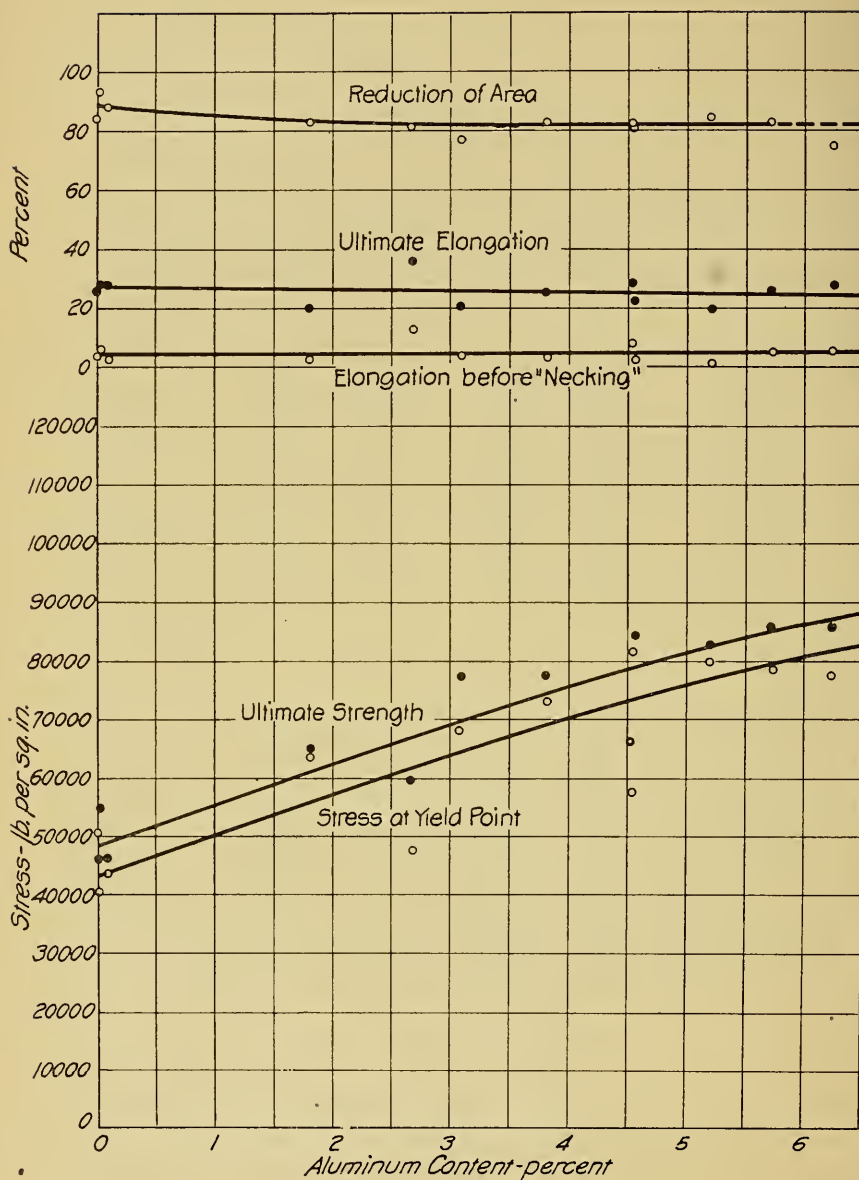


FIG. 3. MECHANICAL PROPERTIES OF IRON-ALUMINUM ALLOYS MELTED IN VACUO.
AS FORGED

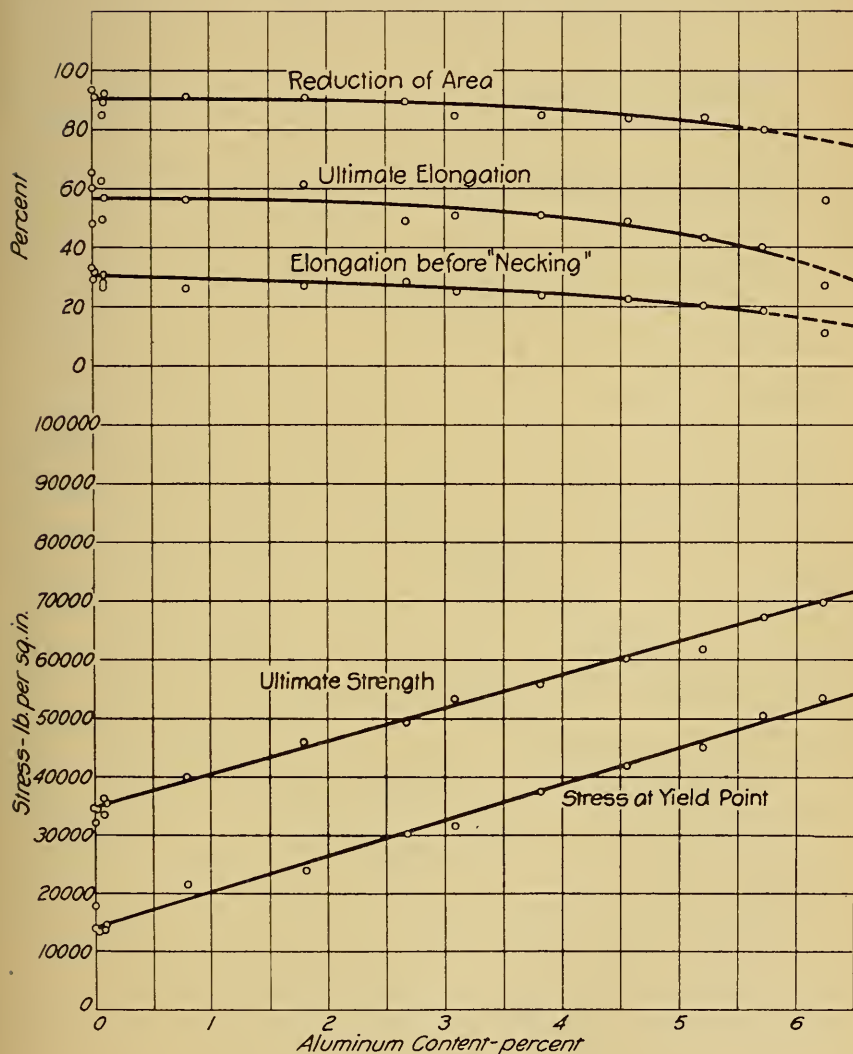


FIG. 4. MECHANICAL PROPERTIES OF IRON-ALUMINUM ALLOYS MELTED IN VACUO. ANNEALED AT 1000 DEGREES C.

some unaccountable reason, too low on comparison with the aluminum added. If the carbon introduced is in proportion to the aluminum added, then the carbon content of these alloys was quite out of proportion to the aluminum content.

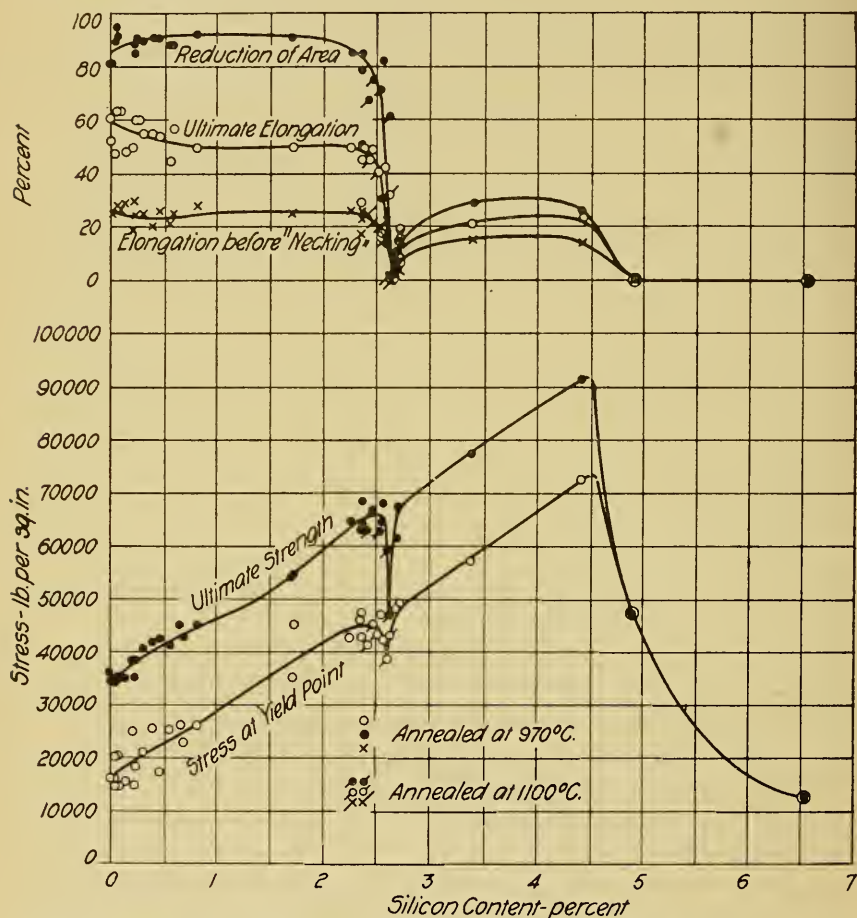


FIG. 5. MECHANICAL PROPERTIES OF IRON-SILICON ALLOYS MELTED IN VACUO. ANNEALED
(FROM BUL. 83)

As a check on this weeding, a number of these alloys were analyzed for carbon; the results of this analysis are shown in Column 4 of Table 2. For low aluminum alloys a carbon content of 0.015 per cent has been allowed; for high aluminum alloys, 0.025 per cent.

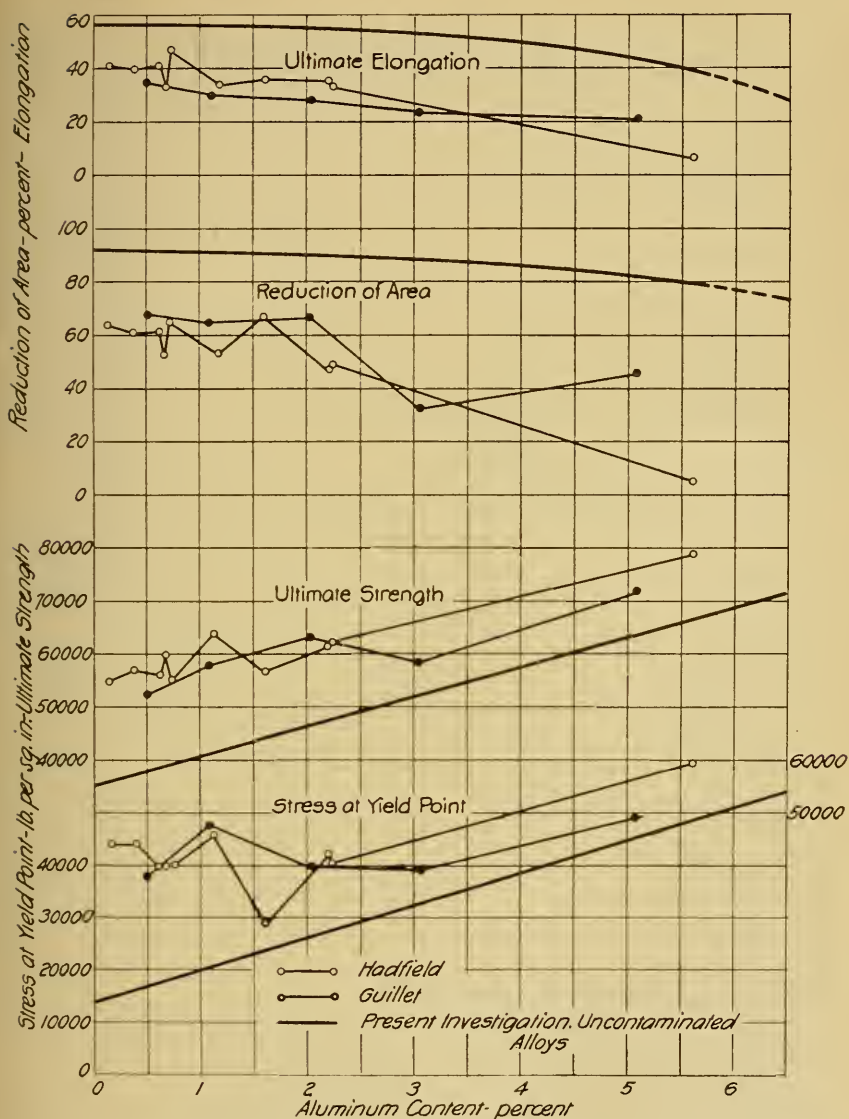


FIG. 6. MECHANICAL PROPERTIES OF IRON-ALUMINUM ALLOYS ACCORDING TO VARIOUS INVESTIGATORS. ANNEALED

IV. RESULTS

1. *Mechanical Properties.*—The results of the mechanical tests are shown in Tables 3 and 4 and in Figs. 3, 4, 7, and 8. For the

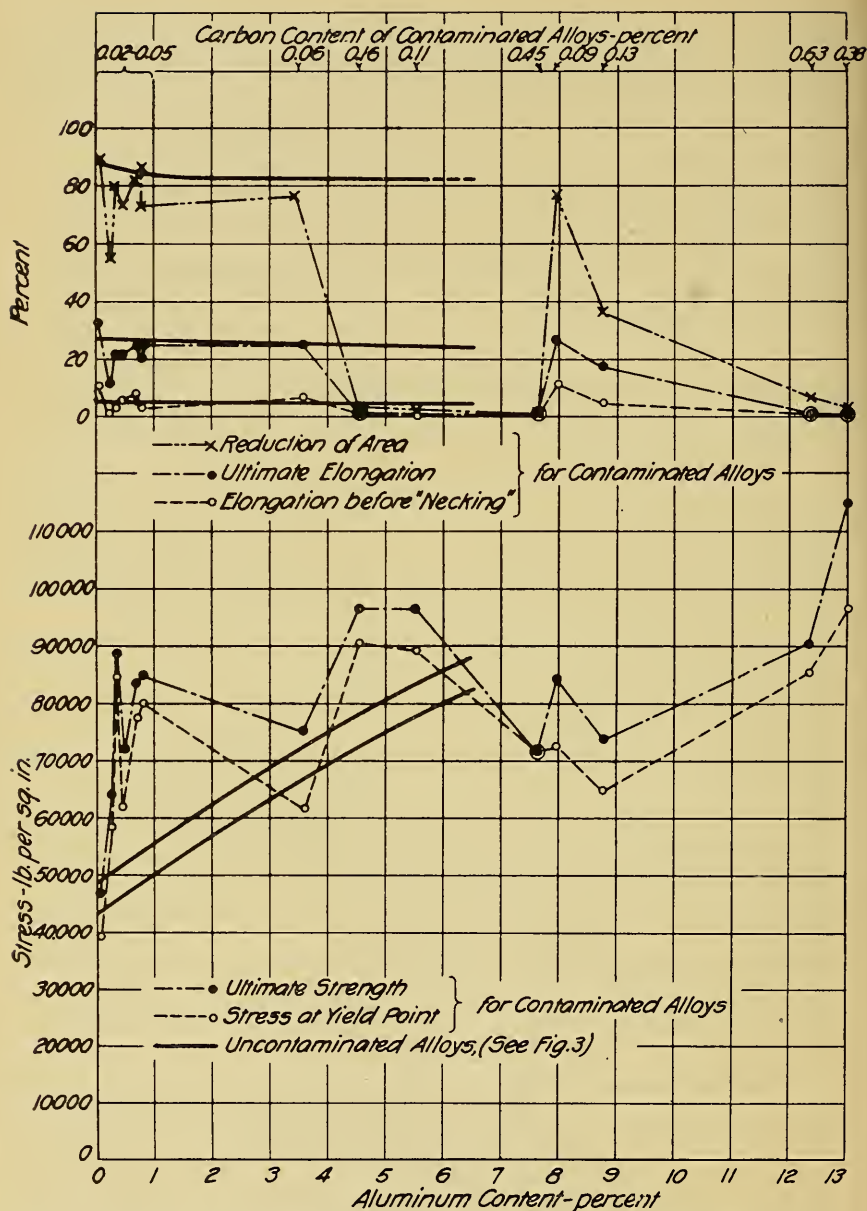


FIG. 7. MECHANICAL PROPERTIES OF IRON-ALUMINUM ALLOYS MELTED IN VACUO, MORE OR LESS CONTAMINATED. COMPARED WITH UNCONTAMINATED ALLOYS. AS FORGED

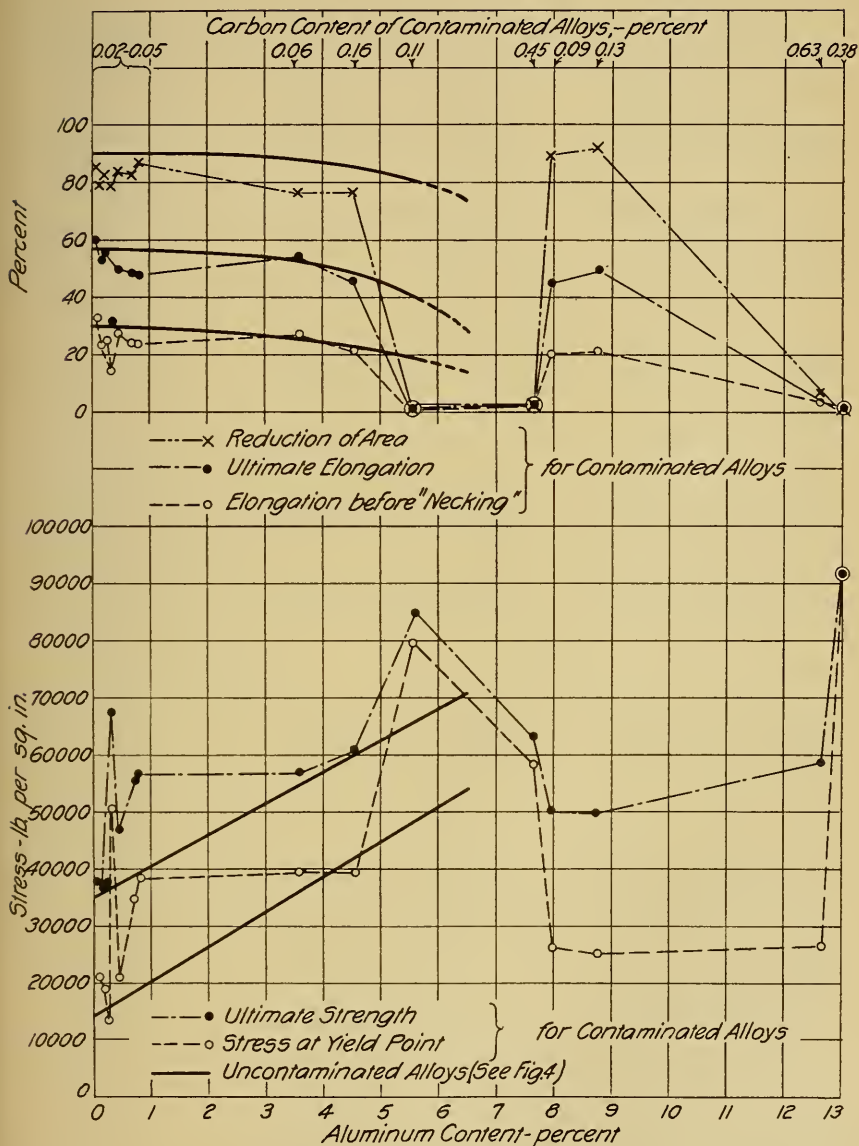


FIG. 8. MECHANICAL PROPERTIES OF IRON-ALUMINUM ALLOYS MELTED IN VACUO, MORE OR LESS CONTAMINATED. COMPARED WITH UNCONTAMINATED ALLOYS. ANNEALED AT 1000 DEGREES C.

sake of comparison, the mechanical properties of the corresponding iron-silicon alloys are shown in Fig. 5.* In Fig. 6 the results of the present investigation are compared with those obtained by Hadfield† and Guillet.‡ Fig. 9 is a photograph of some of the mechanical test pieces after testing.

From the data thus presented, it may be stated in general that aluminum in the absence of carbon increases the strength of iron in almost direct proportion to the amount added. Furthermore, aluminum appears to affect the toughness of iron only slightly. To what aluminum content these rules can be applied is somewhat uncertain, as no alloy containing more than about 6 per cent aluminum uncontaminated by carbon was obtained. How carbon affects these properties may be judged from Figs. 7 and 8. It is apparent that for alloys containing in the neighborhood of 0.1 per cent carbon, aluminum, up to at least 8 per cent, has no marked effect either upon the strength or upon the toughness of iron. This is especially evident in the case of the annealed alloys. The effect of small amounts of carbon is particularly great upon nearly pure iron; it increases the strength of iron about 50 per cent. The strength curves for alloys containing small amounts of carbon will, consequently, be nearly horizontal, and will cross the curves for the more nearly carbon-free alloys at 4 to 5 per cent aluminum, where the effect of carbon seems to be very small. In general the effect of small amounts of carbon is to conceal the true effect of aluminum upon the mechanical properties of iron. If the results given here for alloys with a carbon content of about 0.1 per cent are compared with those given by Hadfield and Guillet (see Fig. 6), they will be found to be very similar. They are more like Guillet's results, which represent alloys containing about 0.1 per cent carbon, than Hadfield's, whose alloys contained about 0.2 per cent. For carbon contents of 0.4 to 0.6 per cent the alloys are forgeable up to at least 13 per cent aluminum. This also agrees with the results of Guillet. For such carbon contents, however, the alloys are quite brittle.

If the effect of aluminum (Fig. 4) upon the mechanical properties is compared with the effect of silicon (Fig. 5), it will be seen that, up to 4.5 per cent, silicon increases the strength much more than does aluminum. On the other hand, although 2.5 to 4.5 per cent silicon

*Taken from "Magnetic and Other Properties of Iron-Silicon Alloys Melted in Vacuo." Univ. of Ill. Eng. Exp. Sta., Bul. 83, 1915.

†Journal Iron and Steel Inst., vol. 38, II, pp. 161-230, 1890.

‡Ibid., vol. 70; II, p. 16, 1906. Rev. de Metal, Memoirs, vol. 2, pp. 312-327, 1905.

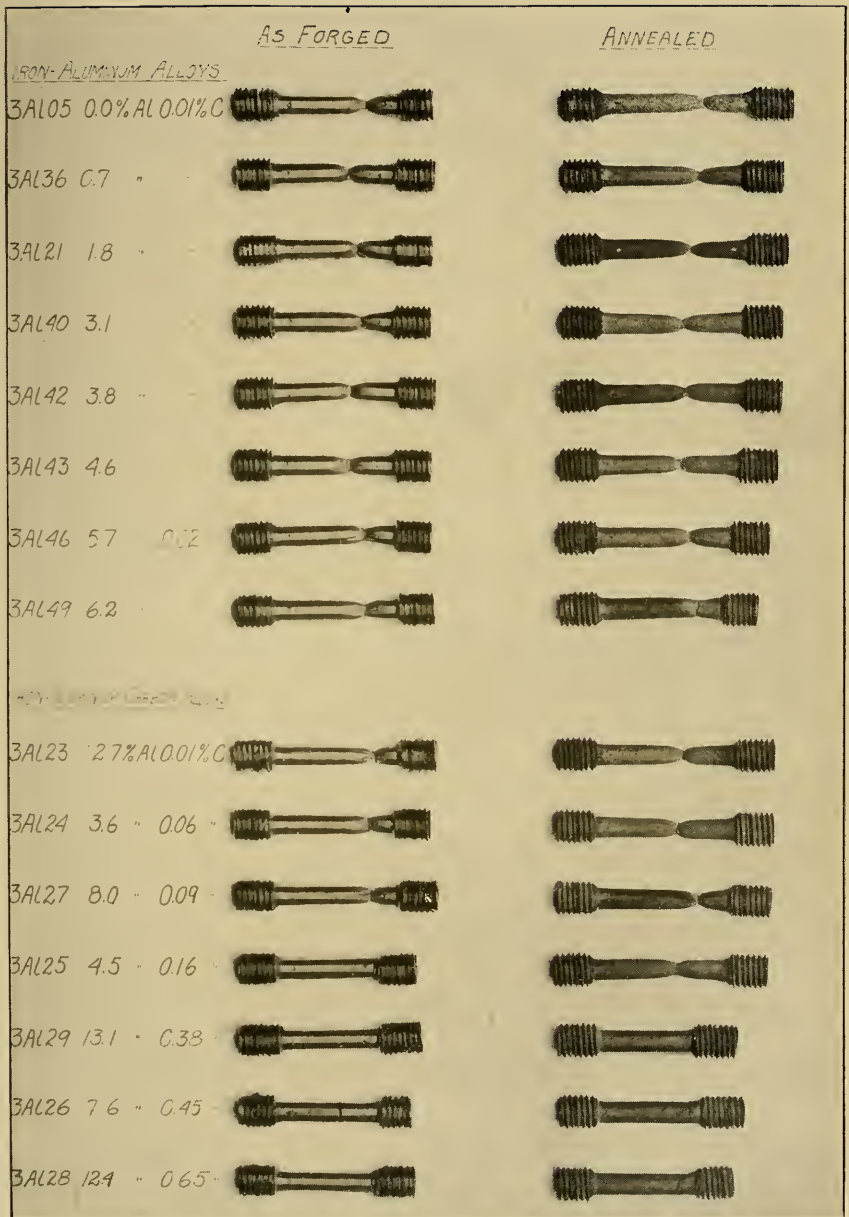


FIG. 9. SOME OF THE MECHANICAL SPECIMENS AFTER BEING TESTED

markedly increases the brittleness of iron, aluminum has no such effect. Furthermore, silicon beyond 4.5 per cent rapidly decreases both the strength and the toughness, while aluminum continues to add strength to the iron without materially affecting the toughness.

2. *Magnetic and Electrical Properties.*—The results of the magnetic and electrical test are shown in Tables 5 to 7 and in Figs.

TABLE 5
RESULTS OF MAGNETIC AND ELECTRICAL TESTS
RODS UNANNEALED

[illegible]

CONTAMINATED ALLOYS

11	0.08		2170	5000	1665	926					...	...	
09	0.09	0.05	2410	4820	1850	943					...	...	
10	0.18		2000	5600	1560	823					...	...	
34	0.22	0.02	2000	6000	1500	528	4400	7630	6200	6700	1.4	1.6	12.7
18	0.23	0.11	1500	6000	1086	550	6480	9600	6290	6760	2.3	2.3	12.6
39	0.32		2280	6600	1886	728	4450	7750	4400	4800	1.5	1.6	14.3
12	0.40		1400	7000	1075	450	7450	14000	6800	7800	2.6	3.1	14.4
35	0.44		2140	6000	1820	833	5090	8400	7800	8200	1.5	1.9	15.3
22	0.45	0.02	1590	6600	1390	667	6360	11000	7105	8160	2.2	2.6	16.1
36	0.69		1930	5600	1430	625	4900	10500	5800	7400	1.7	2.0	18.1
37	0.78		1870	5600	1470	750	4960	8900	6400	7000	1.7	1.9	19.5
24	3.58	0.06	1070	4600	595	281	8270	14000	5300	5363	2.7	2.7	49.5
25	4.52	0.16	573	4300	384	200	12700	21900	5000	5600	4.5	5.1	57.5
51	5.53	0.11	770	3100	370	150					...	...	
27	7.95	0.09	450	3600	202	104	11450	20400	4000	4400	4.7	4.9	78.2
47	8.75	0.13	500	4000	...	...					...	...	

TABLE 6
RESULTS OF MAGNETIC AND ELECTRICAL TESTS
ANNEALED AT 900 DEGREES C.

Rod No.	Aluminum Content Per Cent	Carbon Content Per Cent	Maximum Permeability	Density for Max. Permeability Gauss	Permeability		Hysteresis Loss Ergs per c.c. per Cycle		Retentivity Gauss		Coercive Force Gilberts per cm.		Spec. Elec. Res. at 20° C. Microhms
					For B = 10000	For B = 15000	For B max. = 10000	For B max. = 15000	For B max. = 10000	For B max. = 15000	For B max. = 10000	For B max. = 15000	
3Al.19	0.00		17800	9000	17750	8290	954	2060	9080	14200	.33	.40	9.7
08	0.00		30000	12000	29400	25000	795	1470	9400	14600	.27	.30	9.8
05	0.02		24300	12160	22250	12500	986	2240	9360	14600	.32	.43	10.2
06	0.05		18500	10000	18500	9100	985	2060	9120	14080	.33	.42	10.5
07	0.05		18300	7700	16800	10000	750	1715	8800	12180	.23	.38	10.3
30	0.09		17280	8700	16650	6520	1037	2455	9200	13700	.33	.48	11.4
31	0.09		19500	6400	14500	3840	795	1825	9200	11800	.24	.29	11.2
32	0.10		22500	9000	20800	3950	840	2240	9200	13400	.27	.42	11.2
33	0.37		23500	8000	20400	3750	827	2085	9200	13700	.27	.38	14.1
20	0.80	0.01	14000	8000	12500	3950	878	2458	8600	12750	.35	.47	19.2
21	1.80	0.01	10700	11000	9170	3340	1650	3240	9195	12870	.52	.63	30.7
23	2.67	0.01	12300	8000	11750	1250	1195	2680	8600	11700	.40	.60	39.9
40	3.08		13800	8000	12500	1085	1175	2670	9100	12800	.39	.47	44.3
42	3.82		12700	7000	10500	600	1157	2920	8700	11300	.36	.43	51.6
41	4.54		12360	6800	9270	593	1273	2860	9000	11100	.41	.47	57.6
43	4.55		14600	7200	12560	590	1304	3440	9600	11900	.42	.57	57.3
45	5.20		14680	7200	11500	428	1170	3200	9100	11000	.38	.45	63.0
46	5.72	0.02	11500	6000	7700	273	1290	3150	8600	10200	.41	.45	68.7

CONTAMINATED ALLOYS

11	0.08		13700	8200	13300	6770	1113	2670	8800	13800	.37	.52	11.0
09	0.09	0.05	10750	8600	10500	4840	1227	2740	8520	12200	.37	.50	11.1
10	0.18		11850	9000	11750	5000	1270	2630	8800	12500	.42	.47	12.1
34	0.22	0.02	12000	6000	9100	1610	1080	2730	8300	10600	.33	.43	12.6
39	0.32		20000	10000	20000	6380	902	1905	9400	13200	.30	.40	13.6
35	0.44		15800	10000	14700	5000	1233	2900	9200	14400	.42	.58	15.1
22	0.45	0.02	6500	8000	6060	2140	1930	4590	7990	11380	.35	.45	15.6
36	0.69		18000	9000	17550	3950	985	2170	9100	13700	.33	.41	17.5
37	0.78		12500	7000	11100	3750	1227	2605	8900	12750	.38	.46	18.9
24	3.58	0.06	5630	5200	3570	600	1615	4293	5800	7400	.57	.73	49.2
25	4.52	0.16	2000	4000	1100	341	2490	5530	4300	4700	1.20	1.30	56.8
27	7.95	0.09	3800	3000	1000	150	2350	5080	4500	4550	.80	.80	86.1

TABLE 7
RESULTS OF MAGNETIC AND ELECTRICAL TESTS
ANNEALED AT 1100 DEGREES C.

Rod No.	Aluminum Content Per Cent	Carbon Content Per Cent	Maximum Permeability	Density for Max. Permeability Gauss	Permeability		Hysteresis Loss Ergs per c.c. per Cycle		Retentivity Gauss		Coercive Force Gilberts per cm.		Spec. Elec. Res. at 20° C. Microhms
					For B = 10000	For B = 15000	For B max. = 10000	For B max. = 15000	For B max. = 10000	For B max. = 15000	For B max. = 10000	For B max. = 15000	
19	0.00		17500	7000	16650	7500	890	2100	8835	13000	.30	.41	9.7
08	0.00		22100	6630	20000	7130	700	1370	9400	12200	.22	.26	9.9
05	0.02		20800	6200	18150	7500	699	1780	9195	13000	.23	.35	10.2
06	0.05		20500	10300	20000	10100	780	1971	9200	13710	.26	.43	10.4
07	0.05		30150	9045	30150	12000	670	1620	9400	14200	.22	.33	10.4
30	0.09		23200	6800	17850	5000	610	1261	9075	11180	.19	.22	11.0
31	0.09		26000	5600	17250	4060	522	1080	8520	10200	.16	.19	11.1
32	0.10		34600	7600	30300	6530	427	985	9000	11800	.14	.16	11.6
33	0.37		42700	8400	37600	3750	445	1080	9300	13400	.14	.20	14.1
20	0.80	0.01	20900	7000	17230	3950	700	1590	8920	11800	.23	.25	19.2
50	1.70		22230	6500	18170	2210	670	1560	8900	11800	.21	.25	29.4
21	1.80	0.01	18500	8500	17100	3000	720	1755	9115	13270	.26	.30	30.2
23	2.67	0.01	19700	7500	16150	2000	763	1790	8800	11400	.23	.26	39.7
40	3.08		22100	8600	20000	1150	840	2670	9400	13100	.28	.36	43.9
42	3.82		14600	6000	13150	600	973	2290	8900	11300	.33	.41	51.2
41	4.54		14480	6800	11000	537	1100	2645	8800	10800	.37	.45	57.8
43	4.55		19800	6000	11750	483	985	2530	8800	11300	.27	.52	57.9
45	5.20		21760	6600	11500	375	793	2450	8900	10300	.25	.32	63.5
46	5.72	0.02	12450	6000	7400	273	1210	2300	8800	9900	.38	.43	67.7
49	6.24		16050	3200	7400	259	922	2830	8200	9100	.27	.31	70.4

CONTAMINATED ALLOYS

11	0.08		15000	6500	13900	6300	954	2105	9200	12400	.31	.40	10.9
09	0.09	0.05	11000	8800	10500	4970	1370	3080	9200	12990	.43	.63	11.0
10	0.18		11650	7000	10500	4170	1087	2860	8300	12100	.30	.44	12.0
34	0.22	0.02	18100	5400	11370	1630	706	1423	8000	8500	.20	.20	12.4
39	0.32		20100	7000	17550	7500	890	1843	9500	13000	.29	.35	13.7
35	0.44		16000	9000	15880	5775	1080	2280	9600	14200	.35	.42	15.0
22	0.45	0.02	10200	9000	10000	3000	1270	2960	8400	12590	.42	.52	15.3
36	0.69		20440	9000	20000	5000	1005	2220	9500	14400	.32	.37	18.0
37	0.78		15500	6000	13330	5000	1023	2100	9200	13000	.33	.36	19.0
24	3.58	0.06	6250	5000	3570	600	1390	3330	6100	6800	.45	.53	49.0
25	4.52	0.16	2160	4000	1110	341	2860	5520	4300	4500	1.10	1.20	57.7
51	5.53	0.11	8430	2500	2220	283	1335	3700	4900	5100	.36	.40	66.1
27	7.95	0.09	3060	2300	714	150	2065	2960	3140	3140	.80	.80	81.4

10, 11, 12, 14, and 15. In Fig. 13 the properties of the iron-silicon alloys annealed at 1100 degrees are given for comparison with the corresponding iron-aluminum alloys.

The specific electrical resistance unaffected by any heat treatment, which is about 9.8 microhms for pure iron, is seen to increase by 12 microhms for each per cent aluminum added up to an aluminum content of 3 per cent. For higher values the increase falls off gradually.

Although the electrical resistance is unaffected by heat treatments, the magnetic properties, as is well known, are extremely sensitive to them. From previous investigations on this subject,* it seems probable that the factor which determines the magnetic properties of a certain iron or iron alloy is the internal strain existing in the metal, and that it is the removal of this strain by annealing and slow cooling that is the real cause of any increase in permeability and decrease in hysteresis loss. The transformation of all the β - and γ -iron into α -iron is undoubtedly essential; but in dealing with pure iron, or with iron-aluminum or iron-silicon alloys, this transformation is supposed to take place quite readily, and does not necessitate as slow cooling as 30 degrees per hour. Almost any rate of cooling, short of quenching, seems to effect this transformation, though it has been shown repeatedly that as far as the magnetic properties are concerned, a cooling at the rate of 100 degrees per hour is much inferior to a rate of 30 degrees. A slower rate than 30 degrees per hour does not appear to be of any further advantage. Again, although the transformation from γ - to β -iron (if the β modification actually does exist) takes place at 900 degrees for pure iron, it has been shown that† the most noticeable increase in permeability for low and medium densities takes place upon annealing at between 700 and 800 degrees C. A further, but much smaller, increase takes place upon annealing at 900 degrees. Finally, it was shown in Bulletin No. 83, and it is again shown here, that slow cooling from 1100 degrees produces a higher maximum permeability and a lower hysteresis loss than cooling from slightly above 900 degrees. These facts all favor the theory that, for the kind of iron dealt with in this series of investigations, it is the removal of internal strain from the iron which is of importance in

*"Magnetic and Other Properties of Iron-Silicon Alloys Melted in Vacuo." Univ. of Ill. Eng. Exp. Sta., Bul. 83, pp. 24-27, 1915.

†"Magnetic and Other Properties of Electrolytic Iron Melted in Vacuo." Univ. of Ill. Eng. Exp. Sta., Bul. 72, p. 30, Fig. 3, 1914.

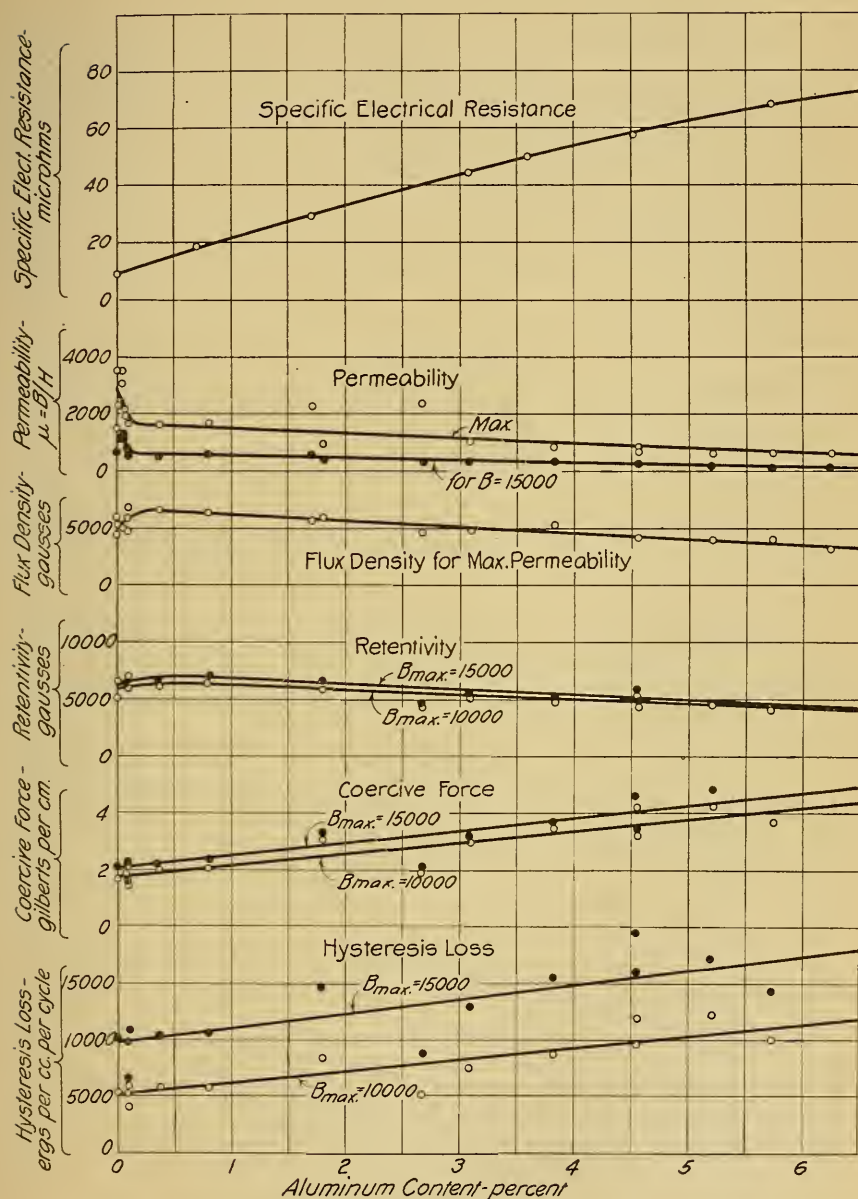


FIG. 10. MAGNETIC AND ELECTRICAL PROPERTIES OF IRON-ALUMINUM ALLOYS MELTED IN VACUO. AS FORGED

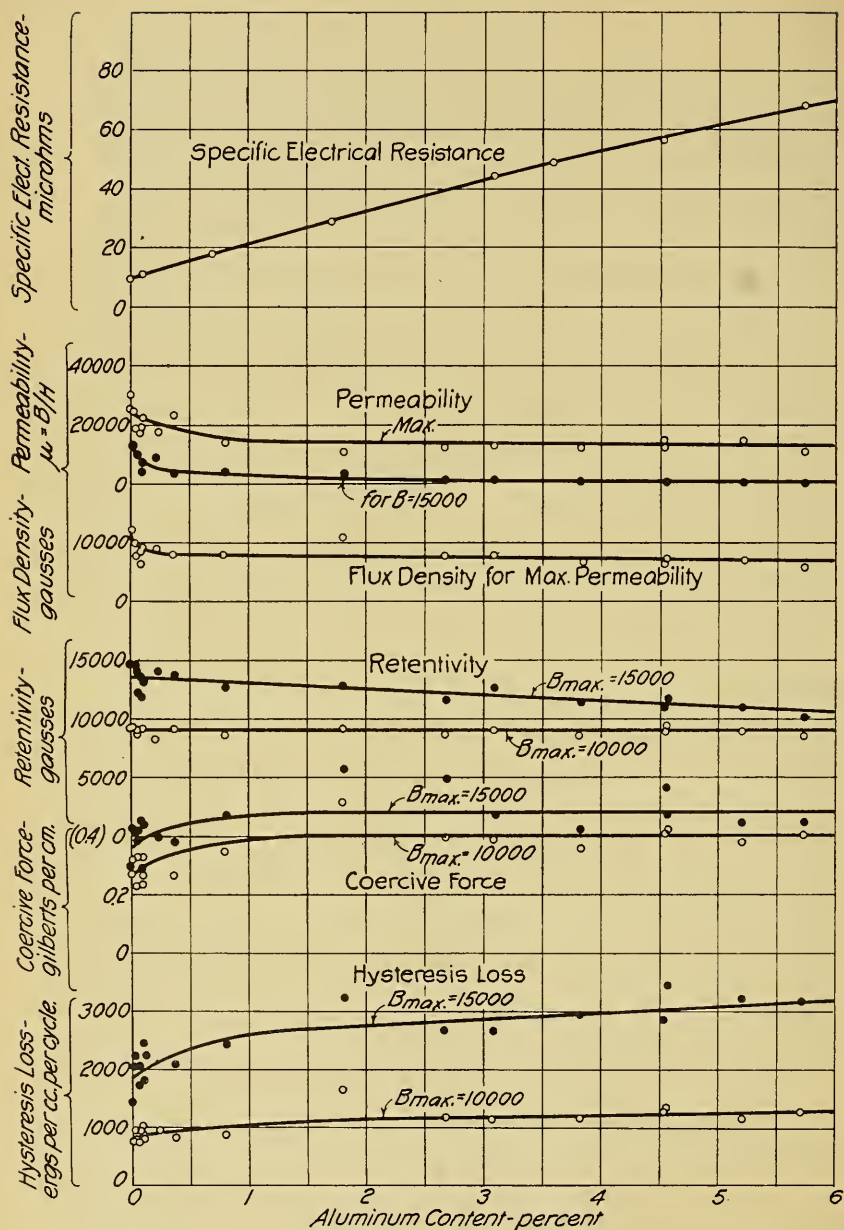


FIG. 11. MAGNETIC AND ELECTRICAL PROPERTIES OF IRON-ALUMINUM ALLOYS MELTED IN VACUO. ANNEALED AT 900 DEGREES C.

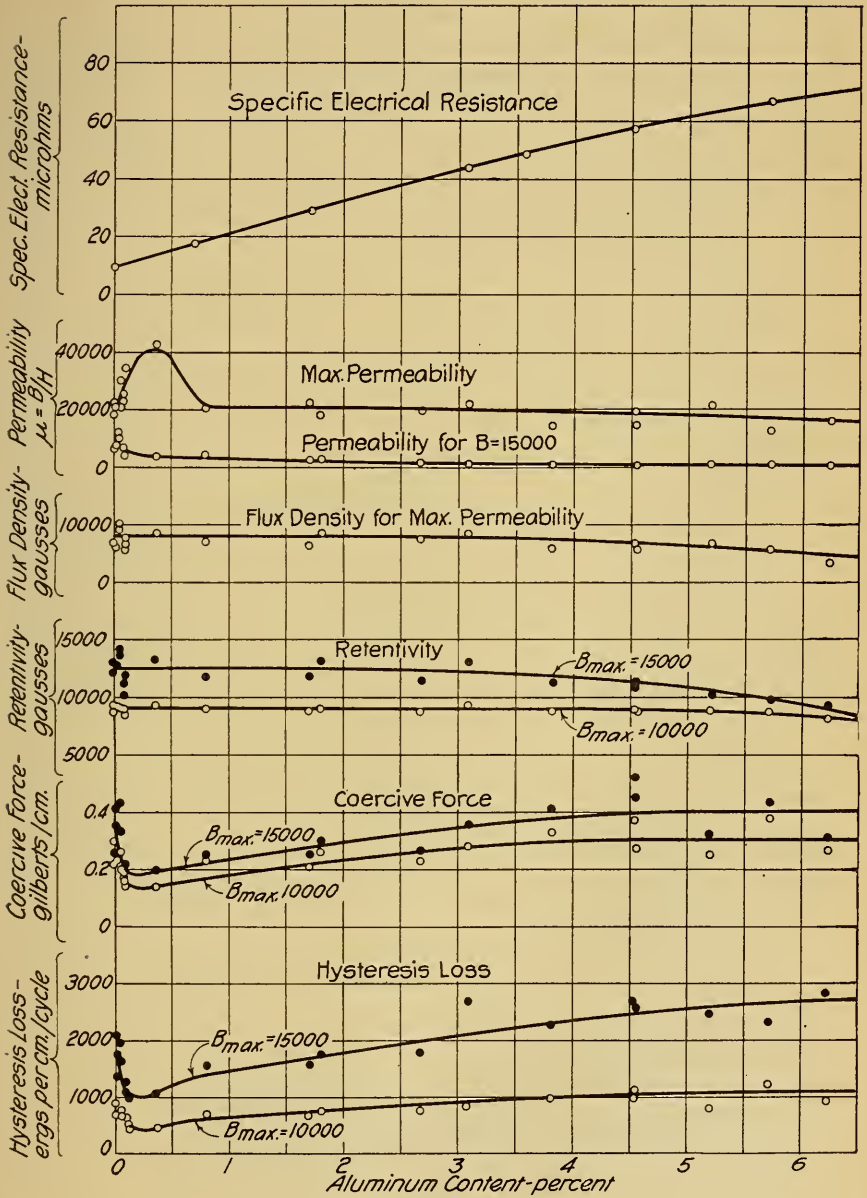


FIG. 12. MAGNETIC AND ELECTRICAL PROPERTIES OF IRON-ALUMINUM ALLOYS MELTED IN VACUO. ANNEALED AT 1100 DEGREES C.

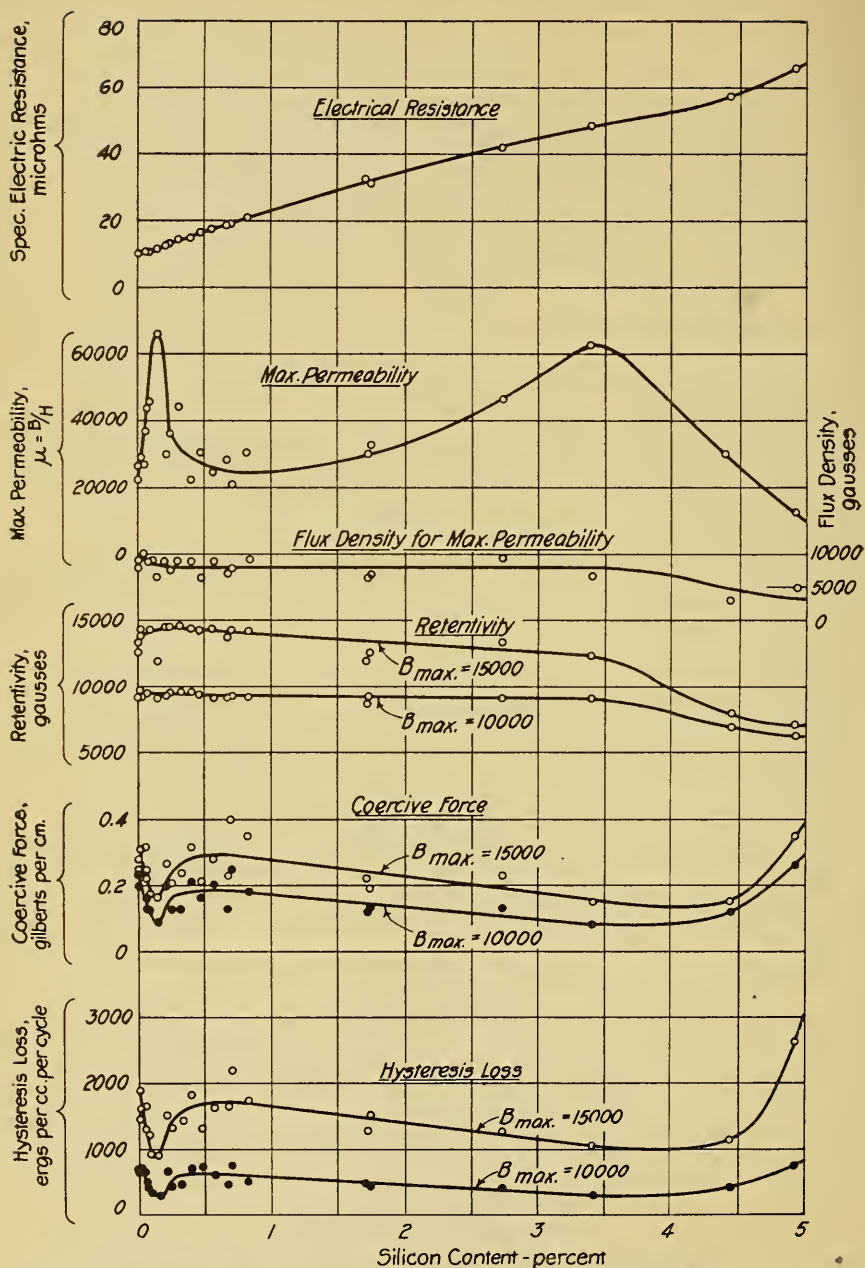


FIG. 13. MAGNETIC AND ELECTRICAL PROPERTIES OF IRON-SILICON ALLOYS MELTED IN VACUO. ANNEALED AT 1100 DEGREES C. (FROM BUL. 83)

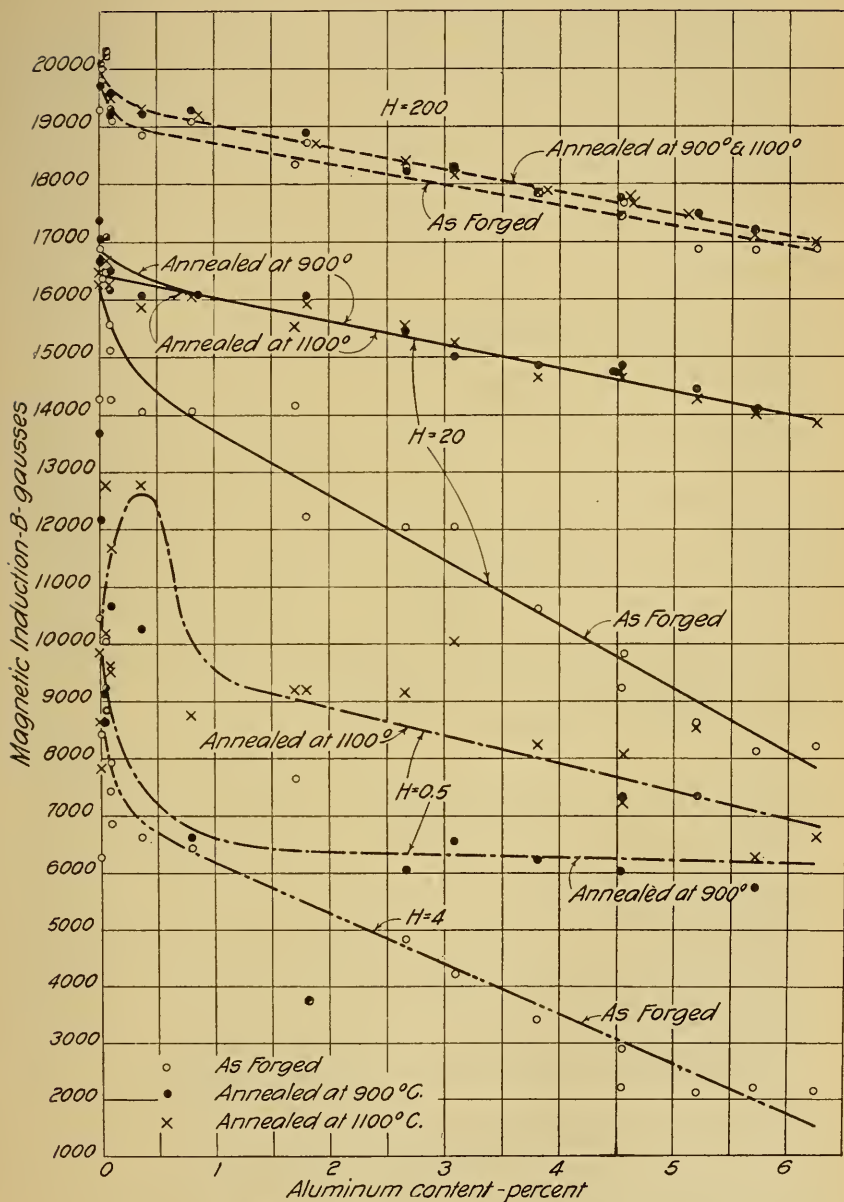


FIG. 14. FLUX DENSITY FOR VARIOUS MAGNETIZING FORCES

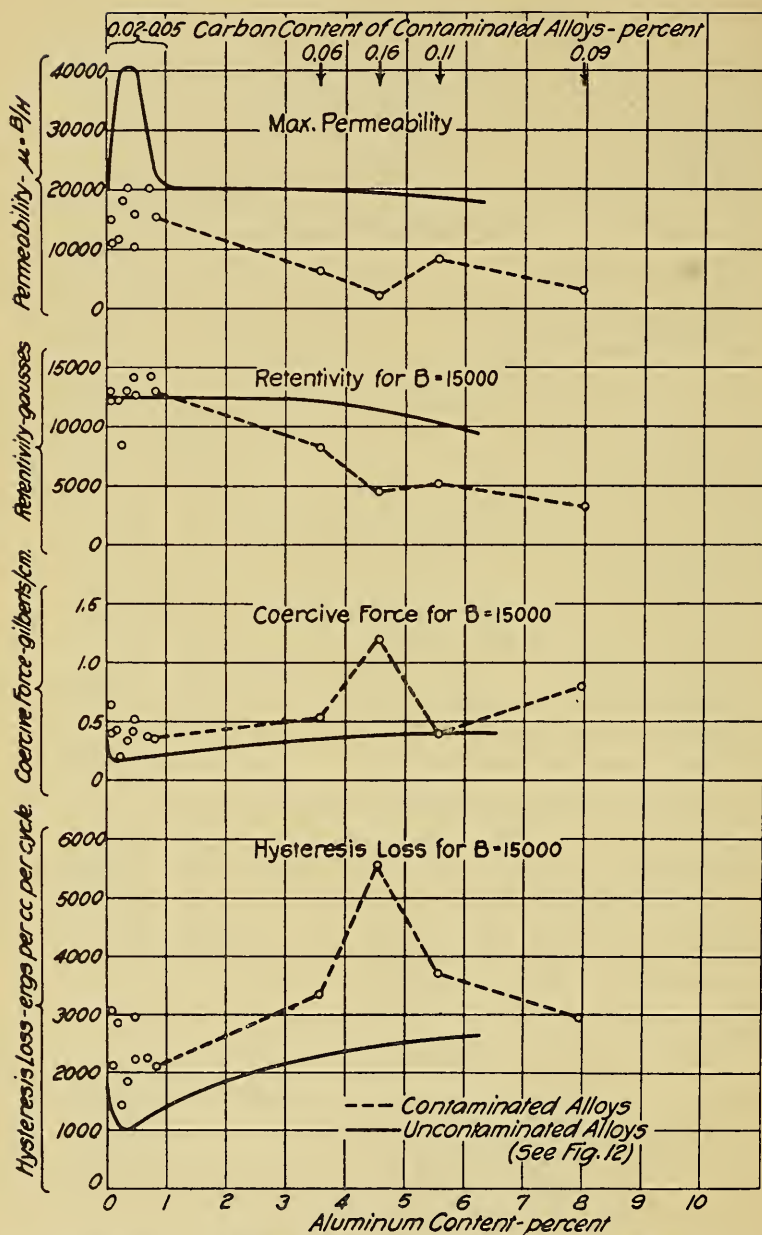


FIG. 15. MAGNETIC PROPERTIES OF IRON-ALUMINUM ALLOYS MELTED IN VACUO, MORE OR LESS CONTAMINATED. COMPARED WITH UNCONTAMINATED ALLOYS. ANNEALED AT 1100 DEGREES C.

the production of high permeability, low hysteresis iron. The transformation of all the iron into α -iron is then a foregone conclusion.

Turning now to Fig. 11, it is seen that the maximum permeability for the unannealed alloys varies from 3500 for pure iron to 600 for a 6 per cent alloy. After the alloys have been annealed at 900 degrees these values (See Fig. 12) increase to 24,000 and 13,000, respectively. The annealing at 1100 degrees (Fig. 13), although not materially affecting the pure iron, raised the maximum permeability of the 6 per cent alloy to 17,000 and that of the 0.4 per cent alloy to about 40,000. The maximum permeability is thus increased ten to thirty times, and the permeability for $B = 15,000$, two to ten times by annealing at 900 degrees. No further improvement takes place by annealing at higher temperatures: in fact, alloys low in aluminum show an actual decrease in annealing at 1100 degrees (see Fig. 14). In regard to the latter point, it has generally been found that if pure iron previously annealed at 900 degrees is annealed at a higher temperature, it decreases in permeability for values of H between 20 and 100 gilberts per cm. For $H = 200$ (Fig. 14) the permeability is the same for the 900- and 1100-degree annealing; the improvement due to annealing amounts to from 1 to 2 per cent. The saturation value is known to be practically unaffected by annealing.*

Annealing lowers the coercive force and the hysteresis loss in much the same ratio as it raises the maximum permeability, the minimum values occurring with an aluminum content of about 0.25 per cent.

When the iron-aluminum and the iron-silicon alloys (Figs. 12 and 13) are compared, the curves, as might be expected, are similar for the low alloys; both have a maximum in the maximum permeability curve and a minimum in the coercive force and hysteresis curves for about 0.2 per cent. The two sets of curves are, however, distinctly different for high alloys. The silicon curves have a second maximum or minimum for from 3.5 to 4 per cent silicon, but the aluminum curves are without such points, the maximum permeability gradually decreasing and the hysteresis loss and coercive force gradually increasing from the 0.5 per cent aluminum point.

Although the iron-aluminum series thus furnishes a high permeability, low hysteresis alloy (similar to the low silicon alloy), it furnishes none which combines these properties with a high electrical resistance in as marked a degree as does the 3.5 per cent iron-silicon

*Gen. Elect. Rev., vol. 18, p. 881. Sept., 1915.

alloy. Nevertheless, the 3.5 per cent iron-aluminum vacuum alloy has a maximum permeability of six times, and a hysteresis loss of only one-half, that of the commercial 3.5 per cent silicon steel. It has been shown, furthermore, that the high aluminum alloys have the advantage over the corresponding silicon alloys of being much tougher mechanically; thus they are more easily worked and lend themselves to some purposes where the silicon alloys are unsuitable on account of their comparative brittleness. The effect of carbon on the magnetic properties is illustrated in Fig. 15, where the maximum permeability, retentivity, coercive force, and hysteresis loss are shown for a number of alloys containing from 0.02 to 0.16 per cent carbon, and are compared with the uncontaminated alloys. The bars have all received the same mechanical and thermal treatment, finally being annealed at 1100 degrees. The maximum permeability attained by the alloys containing about 0.10 per cent carbon is about 5000 as compared with 20,000 for the uncontaminated alloys. The retentivity for these same alloys is much lower, and the coercive force, except in one case, much higher than for the purer alloys. The hysteresis loss is from 50 to 100 per cent and, for the low alloys containing 0.05 per cent carbon, even 200 per cent higher than for the corresponding uncontaminated alloys.

3. *Photomicrographs.*—Photomicrographs of a number of the alloys representative of the iron-aluminum series are shown in Figs. 16 and 17. Fig. 16 shows the uncontaminated alloys and Fig. 17 some of the contaminated ones. On the left hand side are shown the alloys as forged, and on the right the same alloys annealed at 1100 degrees C.

It is clearly seen from these figures that iron and aluminum form a solid solution as has also been shown by previous investigators. In the forged specimens the structure varies greatly, the size of the crystals generally decreasing as the aluminum content increases. The structure of 3 Al 21 is the only exception to this rule, the crystals in this case being unusually large. After the alloys have been annealed at 1100 degrees, this nonuniformity disappears and the alloys exhibit crystals of approximately the same size. For the pure iron, or for the very low alloys, 3 Al 08, 05, and 32, the structure consists, as is usual, of large crystals subdivided into many small ones.*

*See "Magnetic and Other Properties of Electrolytic Iron Melted in Vacuo." Univ. of Ill. Eng. Exp. Sta., Bul. 72, 1914.

"The Effect of Boron upon the Magnetic and Other Properties of Electrolytic Iron Melted in Vacuo." Univ. of Ill. Eng. Exp. Sta., Bul. 77, 1915.

"Magnetic and Other Properties of Iron-Silicon Alloys Melted in Vacuo." Univ. of Ill. Eng. Exp. Sta., Bul. 83, 1915.

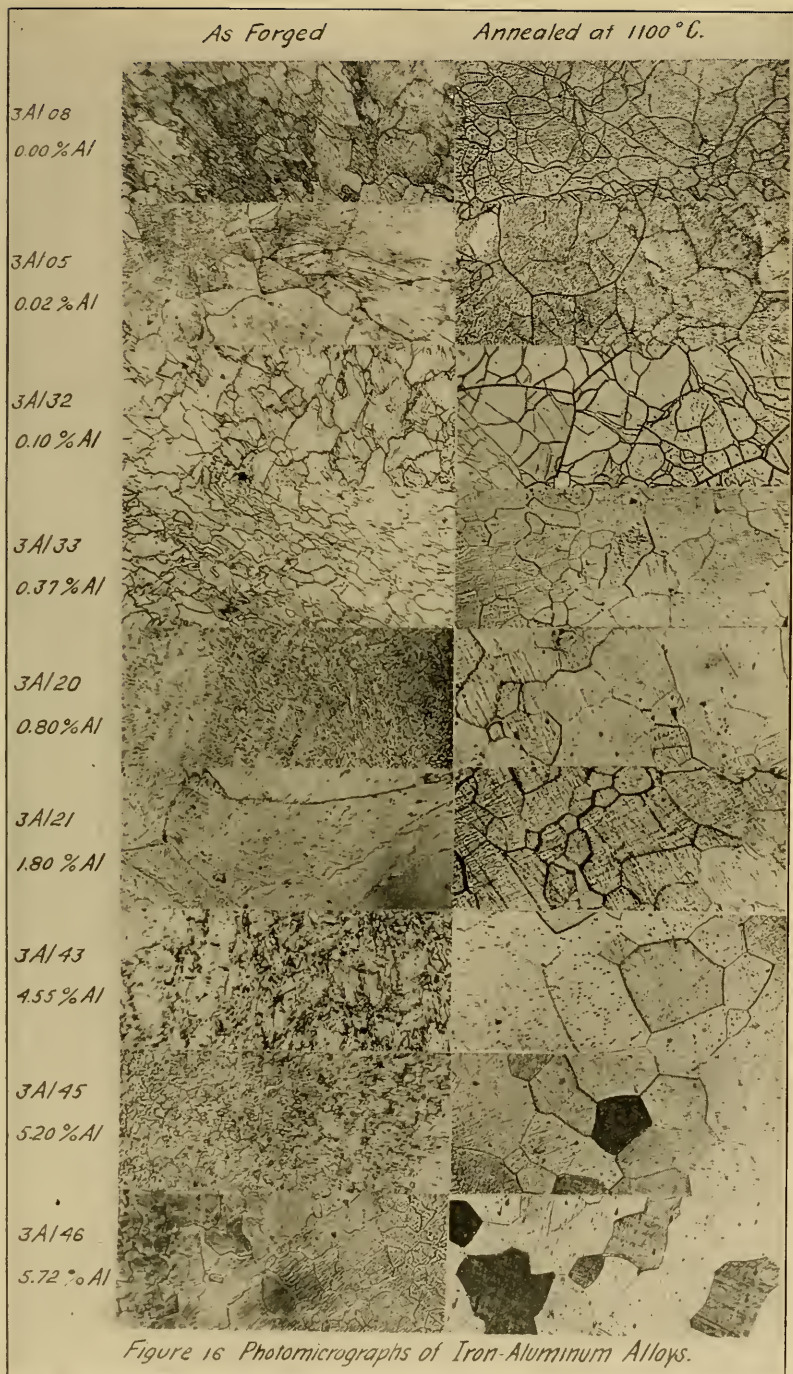


FIG. 16. PHOTOMICROGRAPHS OF IRON-ALUMINUM ALLOYS, AS FORGED AND ANNEALED AT 1100 DEGREES C. $\times 60$

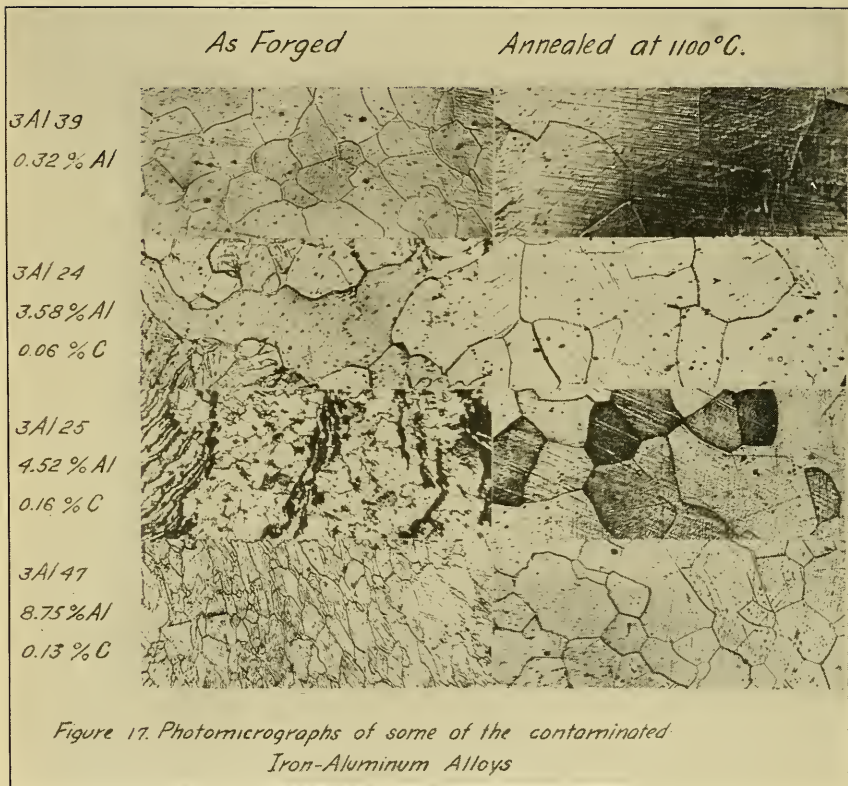


FIG. 17. PHOTOMICROGRAPHS OF SOME OF THE CONTAMINATED IRON-ALUMINUM ALLOYS, AS FORGED AND ANNEALED AT 1100 DEGREES C. $\times 60$

For aluminum contents of 0.40 per cent and above, this subdivided structure does not appear, and the remainder of the alloys are made up of more or less regular crystals.

In the structure of the uncontaminated alloys there is no sign of impurities present; the spots that appear are evidently due to imperfect polishing. In the contaminated specimens the presence of graphite is evident only in the forged specimen of 3 Al 25, containing 0.16 per cent carbon. After annealing at 1100 degrees this evidence of graphite entirely disappeared. What becomes of the carbon in this case has not been ascertained. According to previous investigators carbon in the presence of 4.5 per cent aluminum should be completely precipitated as graphite; but, from the appearance of 3 Al 25 annealed, one is led to believe that the carbon has combined with the iron and aluminum, unless it is precipitated in such finely divided state that it is invisible under the magnification used. It is hoped that this point may be cleared by further investigation.

V. SUMMARY AND CONCLUSIONS

The results recorded in the previous pages may be summarized as follows:

a. The iron-aluminum alloys used in this investigation, prepared by the vacuum method, are less contaminated by impurities than alloys used by previous investigators. The alloys classed as uncontaminated contain only 0.01 to 0.02 per cent carbon. Other alloys containing more carbon are classed as contaminated and are used to show the effects of carbon.

b. Aluminum is more powerful as a deoxidizer than is silicon, for it does not commence to combine with iron until all oxides present are reduced. Aluminum forms a solid solution with iron throughout the range studied.

c. The tensile strength of the vacuum alloys increases in direct proportion to the aluminum content up to at least 6 per cent, the ultimate strength of the latter being 85,000 pounds per square inch (60 kg. per sq. mm.) in the unannealed state and 70,000 pounds (50 kg.) in the annealed. The corresponding figures for pure iron are 48,500 pounds (34 kg.) and 35,000 pounds (25 kg.). The toughness is only slightly affected by the aluminum content.

d. With regard to the magnetic properties aluminum, like silicon, has a very beneficial effect when added in small quantities. The

best alloy obtained, containing about 0.40 per cent aluminum annealed at 1100 degrees, has a maximum permeability above 35,000. The hysteresis loss for $B_{\max} = 10,000$ and 15,000 is 450 and 1000 ergs per cc. per cycle, respectively. For higher aluminum contents the magnetic quality decreases gradually, so that the alloy containing 3.5 per cent aluminum has a maximum permeability of 20,000 and a hysteresis loss for the given densities of 1000 and 2200 ergs, respectively. This loss is only one-half that of the 3.5 per cent commercial silicon steel.

c. The specific electrical resistance increases about twelve microhms for each per cent aluminum added. When the aluminum content exceeds 3 per cent, however, the rate of increase falls off gradually.

By the application of the vacuum process to the iron-aluminum series, an alloy, containing about 0.4 per cent aluminum, has been produced which has remarkable magnetic properties. In this respect aluminum acts like silicon. Aluminum, however, unlike silicon, yields no alloy with similar characteristics for higher percentages. On the other hand, the high aluminum alloys have the advantage over the corresponding silicon alloys of being much less brittle; and this characteristic combined with a fairly high permeability, low hysteresis loss, and an electrical resistance equal to that of the silicon alloys, may make aluminum alloys suitable for certain purposes where the silicon alloys can not be used.

APPENDIX

4. *Magnetic Testing with Burrow's Permeameter.*—During the past two years the attainment of permeabilities above 30,000 in vacuum-fused iron-silicon alloys has made the question of accuracy in results of prime importance. There are three effects which may lead to errors: (a) effect of strain, (b) end effect of the permeameter coils, (c) consequent pole effect due to nonuniformity.

a. The effect of strain has been considered in a previous bulletin.* When the rod is properly clamped, this effect is eliminated.

b. The effect upon H of the ends of the various permeameter coils has also been discussed in a previous bulletin.† The Bureau of Standards' correction of the Burrows method for the theoretical end effect, when the compensating current is not more than twice the current in the two main solenoids, is less than 0.1 per cent. In Bulletin No. 83 of the Engineering Experiment Station of the University of Illinois, it is shown that in some cases the current in the compensating circuit may be thirty times that in the main solenoid. In such cases the correction factor is 2.3 per cent; that is, the measured H must be increased by 2.3 per cent.

c. The effect of nonuniformity of the specimen is one for which no correction can be offered. It is of the same nature as the effect due to the ends of the coils, but it depends only upon the homogeneity of the rod, and, therefore, the error introduced cannot be calculated. Any nonuniformity means that lines of force will leave the iron path and complete all or part of the magnetic circuit in air. Wherever a line of force leaves or enters the iron path, a pole is developed. A line that leaves and re-enters the iron causes two poles of opposite sign. The effect of each pole at the center of the test bar varies inversely as the square of the distance from the center. Hence each pair of poles due to a leakage line has a resultant effect at the center, which depends upon the distance from the center of the points of leaving and of entering the iron. This may be illustrated by the leakage that occurs at the joint of the bar and the yoke. If the lines

*"Magnetic and Other Properties of Iron-Silicon Alloys Melted in Vacuo." Univ. of Ill. Eng. Exp. Sta., Bul. 83, p. 24, 1915.

†"Magnetic and Other Properties of Electrolytic Iron Melted in Vacuo." Univ. of Ill. Eng. Exp. Sta., Bul. 72, p. 61, 1915.

of force leave the rod close to the joint and enter the yoke just across the air gap, the resultant effect on the center of the bar will be small for two reasons: (a) the distance from the center of the resultant pole is great, and (b) the two poles tend to neutralize each other. On the other hand, if the leakage is close to the center of the bar, and the distance is great between the points of leaving the iron and re-entering it, the magnetizing or demagnetizing effect may be great. Leakage of this kind leads to a high percentage of error and is caused by the nonuniformity of the iron. A section of the iron having low reluctance is called a *soft spot*; a section having high reluctance is called a *hard spot*.

In order to investigate the uniformity of rods, a special permeameter was constructed having movable test coils on the outside of the main solenoids. There was only one layer of winding on the solenoids and the maximum value of H obtainable without heating the coils was 60 gilberts per cm. By properly connecting the test coils it was possible to investigate the density at points along the bars and to calculate the amount of leakage at any point at any density up to that corresponding to $H = 60$.

Fig. 18 shows the results obtained with Rod No. 3 Si 27C. For each value of H the leakage along the bar was measured and plotted with the density at the center as the reference point. It should be noted that the diameter of this rod varies as much as 1 per cent, and that at densities above 15,000 the distribution of flux follows the variations in diameter. At lower densities, however, the uniformity of the material has the greater effect, because the iron is more sensitive at densities falling on the steep part of the magnetization curve. From Fig. 18 it can be seen that the assumption that compensation neutralizes only the demagnetizing effect of the leakage at the joints is entirely without justification. In this case a strong magnetizing effect is introduced because of a soft spot about four inches from the center of the rod. If the test coils marked c-c, which are wound over each end of the test bar, were placed three inches from the center they would indicate that a negative current was required for compensation. Such a current would increase the leakage at the joints and would plainly lead to error. If the coils c-c were placed five and one-half inches from the center or close to the yokes, they would then indicate that a positive current was required. However, in raising the ends of the leakage curve, the flux at the point $+4$ would be raised still higher

than for the uncompensated case and the effect would be to increase H at the center beyond its indicated value. When the two coils, c-c, are connected in series and opposed to the test coil at the center, a balance indicates that the average flux at the two ends is equal to that at the

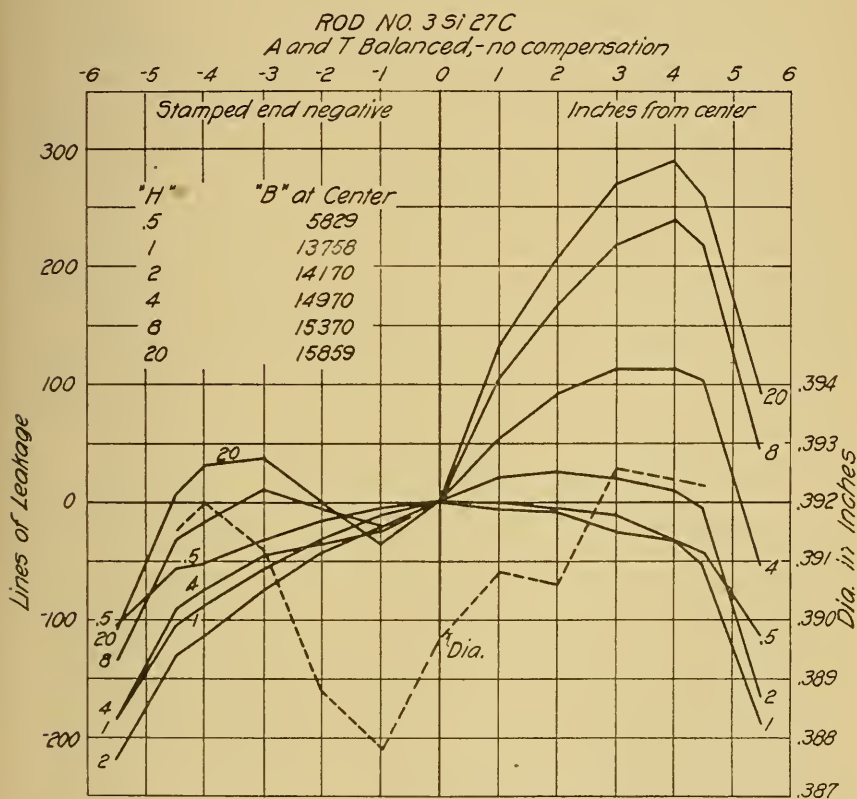


FIG. 18. CURVES SHOWING DISTRIBUTION OF FLUX. NO COMPENSATION.
REFERENCE POINT AT CENTER

center, even though the flux at one end may be much higher than that at the center and the flux at the other end correspondingly lower. Since when the c-c coils were placed half way between the center and the ends, a negative compensating current was sometimes required, it was decided preferable to place them nearer the ends.

Fig. 19 shows the effect of compensation on the value of H at the

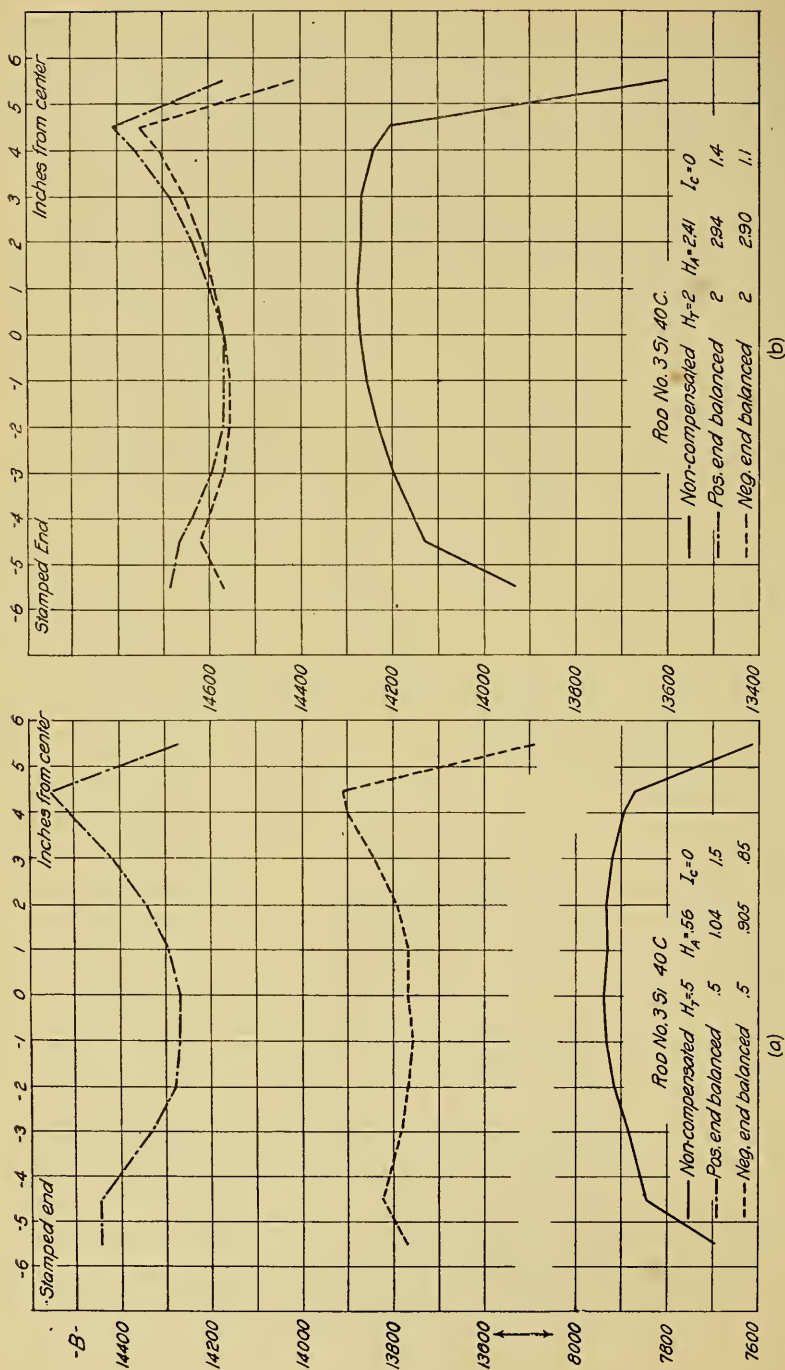


FIG. 19. CURVES SHOWING EFFECT OF COMPENSATION ON FLUX DISTRIBUTION

center for Rod 3 Si 40C. The diameter of this rod is very uniform and the nonuniform flux distribution is due to soft spots near the ends. Two conditions are shown for two values of H . In (a) H is 0.5, and in (b) H is 2.0. The test coils were connected so that the flux at either end of the test bar could be balanced against the center. The solid curve in both figures shows the distribution when the rod is uncompensated. Since the leakage is less for the negative than for the positive end, a smaller compensating current was required at the negative end in order to raise the flux to an equality with the center. The two ends were balanced separately, and the distribution for the two cases is shown by the two dashed curves. Compensation for the positive and negative ends raised the density at the center from 7,870 to 14,280 and 13,680, respectively, for $H = 0.5$; and to 14,280 and 14,580, respectively, for $H = 2.0$. An increase was desired after the leakage at the joints had been neutralized, but there was no way of knowing how much magnetizing effect the "soft spot" had on the center. The density at the center is probably exaggerated, especially for $H = 0.5$.

The double bar and yoke method is based upon the assumption that the rods are uniform. If the rods are uniform, when the c-c coils are placed about half way between the center and the ends, a balance of the average of the flux between these points and the center will result. There may still be some leakage near the joints, but for reasons stated previously its effect is small. As shown in Fig. 18, the distribution can not be assumed to be uniform, and so results by this method are likely to be inconsistent. Nonuniformity is most troublesome in rods of high permeability; with rods of ordinary quality this effect is not so important. For commercial work where only commercial iron is tested, this method is probably sufficiently accurate; but for laboratory testing of high permeability iron, the limiting accuracy should be carefully considered.

5. *Results Obtained with Rings.*—In Table 8 will be found results obtained with rings made of pure open hearth iron, remelted in vacuo. For the sake of comparison, the results obtained with some of the same iron in the form of rods tested by the Burrows method have been included. All the specimens have been annealed at 1100 degrees C. From these results it is seen that for permeabilities not exceeding 20,000, when the possible variation in

mechanical and heat treatment of the various specimens are considered, the two methods check very well.

TABLE 8
COMPARISON BETWEEN THE RING METHOD AND BURROW'S METHOD
OF MAGNETIC TESTING

Specimen No.	Kind of Specimen	Max. Permeability	Density for Max. Permeability	Permeability		Hysteresis Loss Ergs per c.c. per Cycle		Retentivity Gauss		Coercive Force Gilberts per cm.		Remarks
				For B = 10000	For B = 15000	For B = 10000	For B = 15000	For B = 10000	For B = 15000	For B = 10000	For B = 15000	
Open Hearth Iron Remelted in Vacuo and Annealed at 1100 Degrees C.												
4-01	Ring	14300	8500	13700	5700	986	2063	8400	12300	0.33	0.39	
4-01	Rod	14180	8500	13200	5350	1080	2190	8700	12300	0.37	0.40	
4-02	Ring	16500	9500	16450	6400	935	2010	8700	13900	0.30	0.35	
4-03	Ring	17000	9000	16700	8250	852	1755	8400	12600	0.28	0.33	
4-03	Rod	20900	9000	20200	7500	865	1760	9300	13600	0.30	0.34	
4-04	Ring	16300	10000	16300	6000	870	1880	8400	13300	0.30	0.35	
Iron-Aluminum Alloys Melted in Vacuo and Annealed at 1100 Degrees C.												
3 Al 22	Ring	20500	7000	16700	900	636	1320	8400	11000	0.19	0.20	Al-0.45%
3 Al 18	Ring	19700	6400	12500	350	674	1210	9000	10200	0.20	0.20	Al-0.23%

6. *Determination of Aluminum in Iron, by J. M. Lindgren.*—

Four well known methods for the determination of aluminum in iron were tried. First, precipitation by means of sodium hydroxide; second, fusion in a nickel crucible with sodium hydroxide; third, the electrolytic separation with a mercury cathode; and finally, the method of Rothe,* with ether as a solvent for ferric chloride. The first two methods were objectionable because of the slow, tedious filtrations and the resulting voluminous filtrates. The electrolytic method did not remove all the iron, and so made a determination of that constituent necessary. Here, also, tedious filtrations and voluminous filtrates were encountered. Gooch and Hoven† made use of ether in the determination of aluminum by precipitating $\text{AlCl}_3 \cdot 6\text{H}_2\text{O}$ with gaseous hydrochloric acid and completing the separation from iron by means of ether.

Rothe demonstrated that by means of ether large amounts of iron could be separated from nickel-aluminum, copper-cobalt, vanadium, and titanium. He employed a special apparatus for the

*Chem. News, vol. 66, p. 182, 1893.

†Chem. News, vol. 74, p. 296, 1896.

separation, which consisted of two separatory funnels joined to a common stem with a 3-way stopcock. By means of pressure applied by the vapors of ether the solution was forced into the second funnel, where the separation was continued by means of fresh ether. Rothe demonstrated that the iron should be present in the ferric state and that the hydrochloric acid present should have a definite strength.

In the present work the separation was carried out in a 200 cc. Jena beaker. It was found necessary, in transferring from one separatory funnel to another during the process of removing all the iron, to use considerable amounts of hydrochloric acid in order to wash the syrupy aluminum chloride from the sides of the funnel; this increased the acid concentration, and made it very difficult to remove all the iron. It occurred to the writer that the separation could be made easier in the original beaker in which solution was effected by decanting the supernatant ether and ferric chloride, and so make possible a cleaner and easier separation by subsequent washings with ether in the same beaker. The decantation was more easy

TABLE 9
DETERMINATION OF ALUMINUM IN IRON

Per Cent Aluminum Taken	Per Cent Aluminum Found
.732	.763
.732	.791
1.464	1.484

to accomplish than first appeared possible, because the supernatant liquid of ferric chloride and ether was much lighter than the lower solution of aluminum chloride and acid.

Details of the method used are as follows: Dissolve from 1 to 5 grams of iron in a 200 cc. Jena beaker by means of hydrochloric acid, with additions of nitric acid to afford complete oxidation of the iron; evaporate to dryness on a water bath, and take up with just enough hydrochloric acid, sp. gr. 1.19, to completely dissolve the iron. Care should be taken that no iron oxide remain undissolved. Evaporate again on a water bath to a syrup consistency, add 30 cc. of ether, and stir vigorously with a glass rod until the ether has dissolved as much ferric chloride as possible. Allow to stand until the separation is distinct and both layers are clear. At this point it is sometimes necessary to add more hydrochloric acid, two cc. at a time, until the

solutions become perfectly clear. Carefully decant the supernatant solution of iron and ether and repeat the addition of ether. Usually six additions of ether are sufficient to extract completely all of the iron. Evaporate the ether and precipitate the aluminum in the usual way by means of ammonia. Burn the oxide in a porcelain crucible and finally in a muffle at 1000 degrees C to constant weight. The necessity of a high temperature in burning the aluminum precipitate shows the impossibility of obtaining satisfactory results by any of the methods involving the burning off of a mixed precipitate of aluminum and iron.

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UNIVERSITY OF ILLINOIS BULLETIN

ISSUED WEEKLY

Vol. XIV

APRIL 30, 1917

No. 35

[Entered as second-class matter Dec. 11, 1912, at the Post Office at Urbana, Ill., under the Act of Aug. 24, 1912.]

THE EFFECT OF MOUTHPIECES ON THE FLOW OF WATER THROUGH A SUBMERGED SHORT PIPE

BY
FRED B. SEELY



BULLETIN No. 96

ENGINEERING EXPERIMENT STATION

PUBLISHED BY THE UNIVERSITY OF ILLINOIS, URBANA

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BULLETIN No. 96

APRIL, 1917

THE EFFECT OF MOUTHPIECES ON THE
FLOW OF WATER THROUGH A
SUBMERGED SHORT PIPE

BY

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ENGINEERING EXPERIMENT STATION

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THE EFFECT OF MOUTHPIECES ON THE FLOW OF WATER THROUGH A SUBMERGED SHORT PIPE

I. INTRODUCTION

1. *Preliminary.*—This bulletin presents the results of experiments on the flow of water through a submerged short pipe with and without entrance and discharge mouthpieces of a variety of angles and lengths. It treats of the loss of head which occurs when a stream contracts or expands under differing conditions of flow and emphasizes the marked effect that turbulence of flow may have upon the amount of head lost. The discussions have a direct bearing upon various problems in hydraulic practice which involve the contraction and expansion of a stream in flowing through passages.

Comparatively little experimental work has been done to determine the value of conical mouthpieces of various angles and lengths in reducing the lost head at the entrance to and discharge from a submerged pipe, particularly for mouthpieces of the sizes and proportions comparable with those met in engineering practice. The need for such experiments is, therefore, apparent. The minimizing of the lost head due to the contraction and expansion of a stream may be of considerable importance in a variety of hydraulic problems; for example, the intake to a pipe particularly when the pipe is of short length and of large diameter, the suction and discharge pipes of a low head pump, the reduction or expansion from one pipe to another of different diameter or of different shape, the passages through a large valve, the passages through locomotive water columns, the draft tube to a turbine, the connection from a centrifugal pump to a main, the sluice ways through dams, the slat screens at head gates, culverts and short tunnels, jet pumps, the Boyden diffuser as formerly used for the outward flow turbine, the Venturi meter, the suction and discharge pipes of dredges, and the guide vanes and runner of a turbine.

Losses due to this cause are difficult to estimate and easy to overlook. Even where such losses are in themselves of little consequence as compared with other quantities involved, they may have a considerable influence upon subsequent losses on account of the turbulent

motion started by the contraction or expansion. The efficiency of a drainage pump or other low head pump, for example, may be increased by an entrance mouthpiece on the suction pipe because it allows the pump to receive the water in a smoother condition of flow. It is well known that a turbine must receive the water from the guide vanes without shock if, in the subsequent flow through the runner, the energy of the water is to be absorbed efficiently by the turbine. The loss of head through a Venturi meter may be considerably increased if the meter is placed too short a distance downstream from a valve, elbow, or other obstruction or cause of disturbance in the pipe. The friction factor for a pipe following an obstruction or bend may be changed by the disturbance thus caused; the lost head at the entrance to a pipe, particularly when inward-projecting, may be more than that ordinarily assumed for a tube three diameters long. There is but little definite knowledge on the whole subject of the effect of abnormal conditions, and it offers a large scope for investigation. The fact that a comparatively small change in the form of the blades of a turbine runner may result in a large effect on the efficiency of the turbine should prove suggestive when estimating the probable effect of turbulent flow in less severe or critical cases. It is also worth mentioning in this connection that the recent advances in turbine design have been due largely to the attention given to the approach channels to the guide vanes and to the design of the draft tube.

The flow of water usual in engineering practice is more or less turbulent. The general equation of energy, or Bernoulli's theorem, so generally used in hydraulics, applies only when the particles of water move with uniform velocity in parallel stream-lines. Although this condition of flow seldom occurs, satisfactory analyses may often be made by using an average velocity and introducing empirical constants. However, a very slight change in the conditions under which flow takes place may cause, in some cases, a large difference in the action or behavior of the water. There is, therefore, always danger in extending the use of experimental data or empirical constants to apply to conditions of flow quite different from those under which the data were obtained.

2. *Acknowledgment.*—All the experimenting was done in the hydraulic laboratory of the University of Illinois under the general direction of Professor A. N. Talbot. A part of the problem and some

of the methods had been developed by Professor Talbot through experimental work which had been carried on in the hydraulic laboratory for a number of years. The types and proportions of the mouthpieces and the general features of the apparatus had been planned by him. Thesis work of the following students has been utilized as preliminary material in helping to determine the methods used in the investigation: W. P. Ireland, "Entrance Head in Pipes and Conduits," 1903; C. C. Wiley, "Entrance Head and Discharge Head in Pipes," 1904; W. R. Robinson, "Entrance Head and Discharge Head in Pipes," 1906; W. R. Robinson, "An Investigation of the Flow of Water through Submerged Orifices and Pipes," 1909. Although but few of the data given in these theses were incorporated in the final results as reported in this bulletin, they were of considerable value in determining the influence of certain factors involved in the method of experimenting.

The major part of the experimenting was carried out by the writer during 1914 and 1915, with the help of Mr. L. J. Larsen and Mr. R. L. Templin, research fellows in the Engineering Experiment Station, whose careful work is gratefully acknowledged.

II. APPARATUS AND METHOD OF EXPERIMENTING

3. *Short Pipe*.—The cast-iron short pipe to which the mouthpieces were attached was $22\frac{1}{2}$ in. long, bored to a smooth surface and to a 6-in. diameter; an average of thirty micrometer readings taken across three diameters at each of ten successive sections along the pipe gave 0.5995 in. It was threaded at each end so that a mouthpiece could be screwed on either end or on both ends. Fig. 1 shows the 6-in. short pipe used in the experiments. Fig. 5 is from a photograph and shows the tank used in the experiments and also the 6-in. short pipe and some of the mouthpieces. A flange near the middle of the pipe was used to attach it to a partition separating the two compartments of the tank, allowing the pipe to project into each compartment. Fig. 2 shows the short pipe in place in the tank with mouthpieces attached.

Experiments were also made on a steel tube 3.11 in. in diameter by 12 in. long used as an inward-projecting short pipe only with no mouthpiece attached.

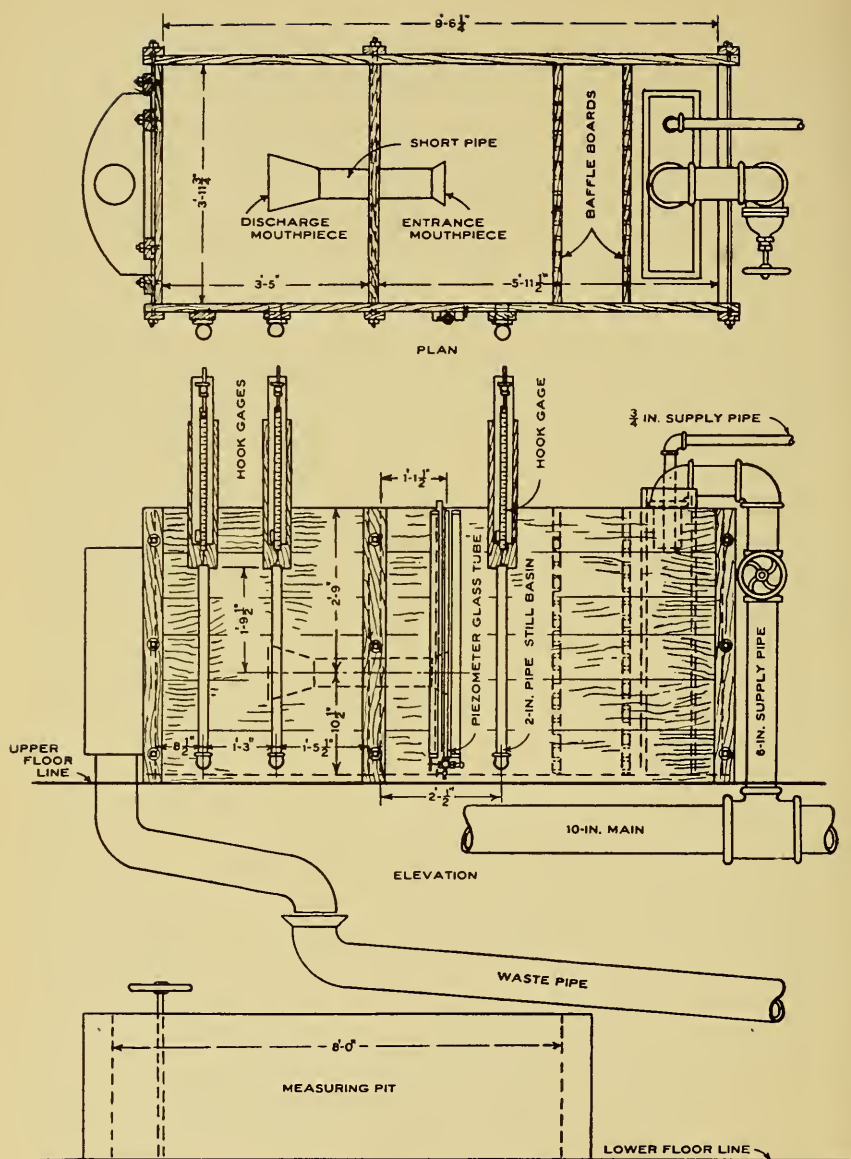


FIG. 2. TANK AND ARRANGEMENT OF APPARATUS

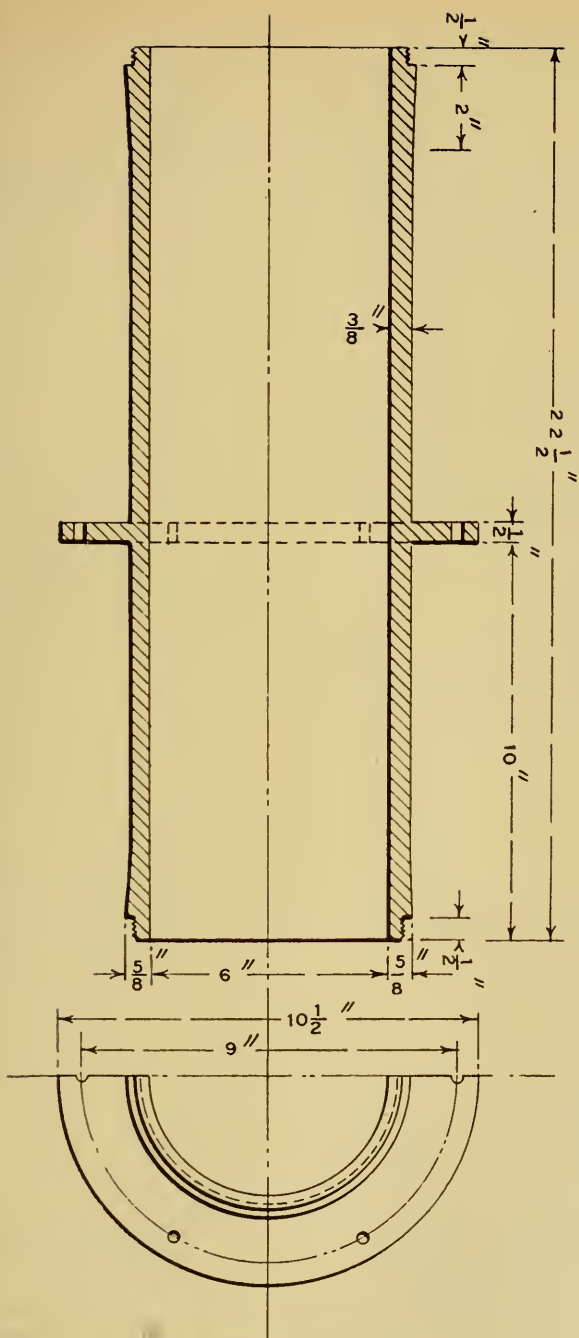
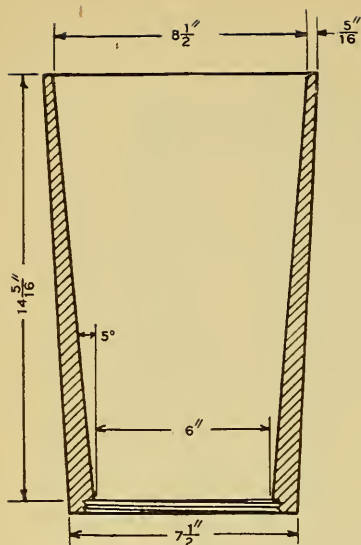
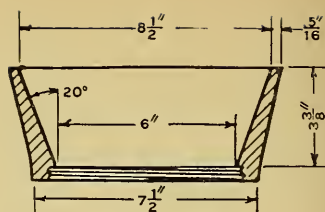


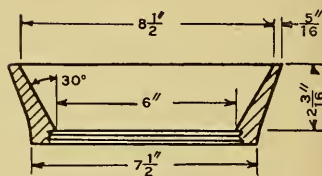
FIG. 1. SHORT PIPE TO WHICH MOUTHPIECES WERE ATTACHED



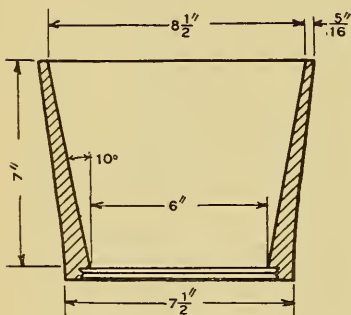
5° MOUTHPIECE



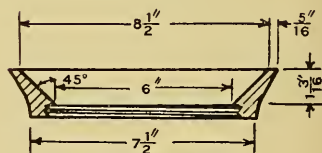
20° MOUTHPIECE



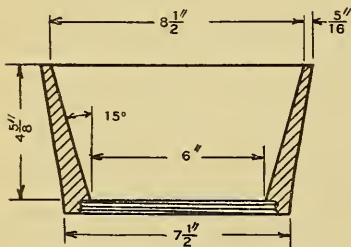
30° MOUTHPIECE



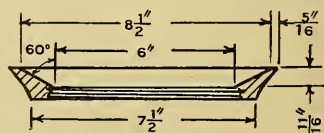
10° MOUTHPIECE



45° MOUTHPIECE



15° MOUTHPIECE



60° MOUTHPIECE

FIG. 3. CONICAL MOUTHPIECES HAVING AN AREA RATIO OF 1 TO 2

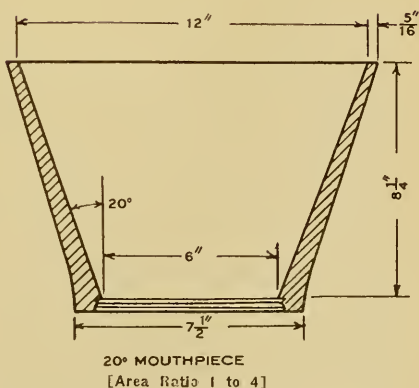
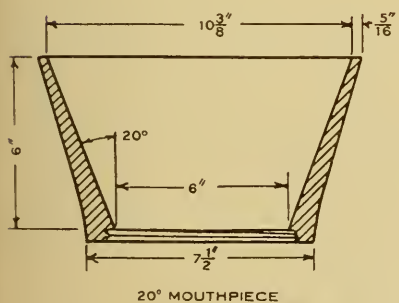
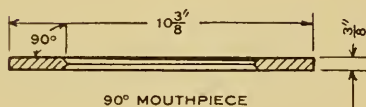
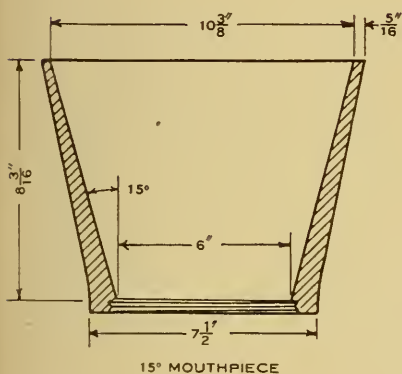
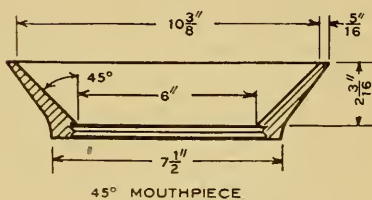
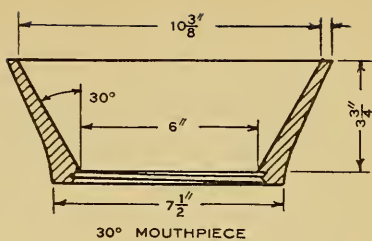
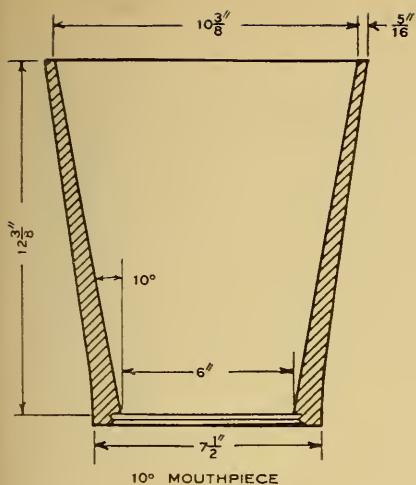


FIG. 4. CONICAL MOUTHPIECES HAVING AN AREA RATIO OF 1 TO 3

4. *Mouthpieces.*—The cast-iron conical mouthpieces which were screwed on the ends of the 6-in. pipe are shown in Figs. 3 and 4. The smallest cross-sectional area of each mouthpiece was the same as the area of the pipe with which the mouthpiece made a smooth connection. The largest or outer area of one series of mouthpieces was twice the area of the pipe, that is, the ratio of the area of the pipe to the largest area of the mouthpiece was 1 to 2. Another series had an area ratio of 1 to 3, and one mouthpiece (20 degree angle) had an area ratio of 1 to 4. The length of a mouthpiece for any area ratio depends, of course, upon the angle of the mouthpiece. Table 1 gives

TABLE 1
LIST OF MOUTHPIECES USED ON THE SHORT PIPE

Each Mouthpiece Tested Singly on Entrance End and Discharge End				Combinations			
Area Ratio	Angle of Mouthpiece Degrees	Area Ratio	Angle of Mouthpiece Degrees	Entrance End		Discharge End	
				Degrees	Area Ratio	Degrees	Area Ratio
1 to 2	5	1 to 3	10	20	1 to 2	5	1 to 2
	10			20	1 to 3	5	1 to 2
	15			20	1 to 2	10	1 to 3
	20			20	1 to 2	10	1 to 2
	30			30	1 to 2	10	1 to 2
	45			20	1 to 2	15	1 to 3
	60			15	1 to 2	15	1 to 2
				20	1 to 2	20	1 to 4
				20	1 to 2	20	1 to 3
				20	1 to 3	20	1 to 2
		1 to 4	20	30	1 to 2	30	1 to 2

a list of the mouthpieces used, together with the particular combinations of an entrance and a discharge (exit or diverging) mouthpiece employed. In no case was a mouthpiece used alone, that is, without being attached to the short pipe. It will be noted that any mouthpiece could be attached to the entrance end of the short pipe only, or to the discharge end only, or two could be attached, one on each end of the pipe. In the case of a few of the mouthpieces, duplicates were made. The angle of the mouthpiece as given in Table 1 and in the various figures, and as used in these pages, means the angle between the axis line of the pipe and one element of the cone, not the total angle of convergence or divergence. Hence, a 90-degree entrance mouthpiece is a flat disc giving a square or flush entrance to the pipe.

5. *Tank and Method of Experimenting.*—The same tank was used in all the experiments. The dimensions of the tank are shown in Fig. 2 and a photograph gives other details in Fig. 5. The tank was divided into two compartments by a vertical partition to which the short pipe was attached in a horizontal position.

Water from the laboratory standpipe was supplied to the tank through a 6-in. supply pipe and also through a $\frac{3}{4}$ -in. pipe, the latter making possible a finer adjustment in maintaining a constant head. After passing through baffle boards the water flowed through the short pipe and finally left the downstream compartment by passing out of the small openings in the end of the tank, the flow through which was regulated by placing stoppers in some of the holes. These holes were arranged in two long vertical rows in the end of the tank, one near each side, and the stoppers were arranged so as to give nearly the same distribution of flow from each row. This, it was found, helped to maintain steady conditions.

The quantity of water discharged was measured in a pit about 6 ft. deep, with a diameter of 7.995 ft. as obtained from readings of a micrometer attached to a rigid stick. The rise in the pit was determined by a vertical graduated rod which was read directly to 0.02 ft. and to 0.004 ft. by estimating. A float was attached to the bottom of the rod and a still basin was provided. The water was wasted into another pit through a movable spout until the surface of the water in the measuring pit became fairly still so that an accurate reading of the rod could be taken. A hook gauge was used to test the accuracy of the float and rod. At the end of the experiment the water was again wasted in the same manner. A calibrated stop watch gave the time corresponding to the rise in the pit.

The head causing flow through the short pipe is the difference in the levels of the water surfaces in the two compartments of the tank. The head was measured in nearly all the experiments by means of hook gauges. These gauges were read directly to 0.001 ft. and to 0.0005 ft. by estimating. Vertical 2-in. pipes attached toward the bottom of the tank served as still basins for the hook gauges (see Figs. 2 and 5). The level of the water in the upstream compartment was determined by the use of one hook gauge only, but two gauges were used on the downstream compartment in the earlier experiments. It was found, however, that for the lower heads the two gauges gave practically the same result and for the higher heads the

gauge nearer the partition gave less fluctuation. For these reasons, and because of less difficulty in getting simultaneous readings of only two gauges, it was decided to take readings with one gauge only on each compartment.

Zero readings of the hook gauge were obtained by reading the gauges when the tank was nearly full and when no water was allowed to escape, the levels of the water surfaces in the two compartments then being the same. Zero readings were taken frequently during the experiments.

For most of the heads above 0.3 ft. the head was measured by a differential gauge which was connected to each compartment of the tank by means of rubber hose. A mixture of carbon tetrachloride and gasoline having a specific gravity of 1.25 was used; thus making the gauge reading four times the head. The gauge was provided with a scale graduated to 0.005 ft. In a few cases two vertical piezometer glasses were used, one attached near the bottom of each compartment, the difference in readings of which (corrected for zero reading) gave the head to 0.001 ft. These three methods overlapped somewhat so that certain heads were measured by all three methods.

Leakage from the tank and from the measuring pit was determined several times during the progress of the experiments and was found to be negligible.

The following procedure comprised an experiment: Stoppers were removed from the end of the tank in sufficient number to give the desired discharge, and the inflow through the 6-in. and $\frac{3}{4}$ -in. pipes was then adjusted until the difference in levels of the water surfaces in the two compartments of the tank became constant. The $\frac{3}{4}$ -in. supply pipe was used to make the final adjustment of the head and to hold the head constant throughout the experiment. After obtaining a constant head, the waste pipe shown in Fig. 2 was pulled from beneath the discharge pipe, allowing the water to discharge into the measuring pit until the rise in the pit was sufficient to allow of its measurement without appreciable error, and to allow an accurate measurement of the head to be made. The head was taken as an average of from two to ten readings of the hook gauges, the larger number being necessary with the higher velocities on account of the greater fluctuations of the water levels due to the more turbulent conditions of the water, especially in the downstream compartment. Each experiment was repeated three times, as a rule, although in the

$$\text{Therefore, } h' = \left(\frac{1}{c^2} - 1 \right) \frac{v^2}{2g} \quad . \quad . \quad . \quad . \quad (3)$$

in which c is the coefficient of discharge since the pipe flows full at the discharge end, and is the ratio of the measured rate of discharge, q , to the theoretical rate of discharge, or, in equation form,

$$c = \frac{q}{a\sqrt{2gh}} \quad . \quad . \quad . \quad . \quad . \quad (4)$$

in which a is the area of the cross-section of the pipe. The expression

$\left(\frac{1}{c^2} - 1 \right)$ is called the coefficient of loss and is denoted by m .

If a converging or entrance mouthpiece is attached to the short pipe, the contraction of the stream will be somewhat suppressed. The discharge, therefore, will be increased, with a corresponding reduction in the lost head. How much of this decrease in energy loss occurs at the entrance and how much during the subsequent flow in the pipe for any given case is difficult to say. The loss of head which would occur in a 6-in. pipe 22½ in. long, if it were a part of a longer pipe and preceded by a considerable length of straight pipe of the same diameter, would be that due to pipe friction,

$$f \frac{1}{d} \frac{v^2}{2g} \text{ or } 0.086 \frac{v^2}{2g}.$$

This loss is not considered in these experiments for, although the pipe may flow full for the greater part of its length when the contraction at entrance is largely suppressed by an entrance mouthpiece, the state of flow is no doubt determined chiefly by the entrance conditions and the loss of head should all be considered as entrance loss.

If a diverging mouthpiece is attached to the discharge end of the short pipe, a part of the velocity head in the pipe may be regained. Theoretically, the amount possible of recovery is the difference between the velocity head in the pipe, $\frac{v^2}{2g}$, and the velocity head at the outer or discharge area of the mouthpiece, $\frac{v_0^2}{2g}$. This may be

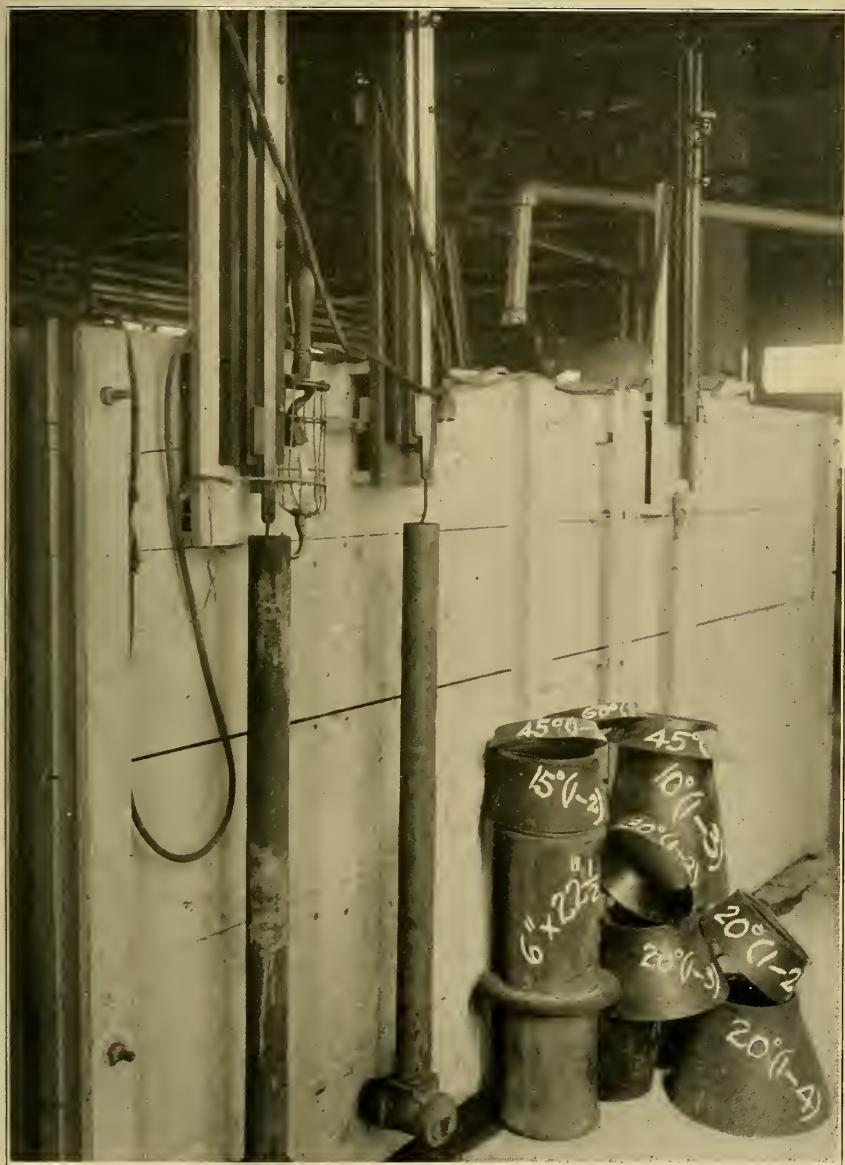


FIG. 5. VIEW OF TANK, OF SHORT PIPE, AND OF SOME OF THE MOUTHPIECES

expressed as

$$\left[1 - \left(\frac{a}{A} \right)^2 \right] \frac{v^2}{2g},$$

in which $\frac{a}{A}$ is the ratio of the area of the pipe to the outer area of the discharge mouthpiece. In other words, if the water approaches the mouthpiece with parallel stream-lines (smooth flow) and if the mouthpiece could reduce the velocity without eddying—ideal conditions—a discharge mouthpiece with an area ratio of 1 to 2 would regain 75 per cent of the velocity head in the pipe, while a mouthpiece having an area ratio of 1 to 3 would regain 88 per cent of the velocity head and a 1 to 4 mouthpiece would recover 94 per cent.

The flow in a short pipe, particularly when no mouthpiece is attached to the entrance end, is far from smooth. And while this turbulence of flow would be expected to change materially the action or effect of the discharge mouthpiece from that occurring under ideal conditions, there is no rational method of taking this into account. It seems probable, however, that the discharge mouthpiece influences the flow for some distance back in the pipe so that the loss of head in the tubes is less than when no discharge mouthpiece is used. This action in turn allows the mouthpiece to act with greater efficiency. This will be discussed later.

It should be noted that in the expression for the coefficient of loss,

$$m = \left(\frac{1}{c^2} - 1 \right) \cdot \cdot \cdot \cdot \cdot \cdot \quad (5)$$

c is the coefficient of discharge based on the area of exit from the system. Hence in the case of the short pipe with a discharge mouthpiece attached, the lost head is expressed in terms of the velocity head at exit from the mouthpiece. But since $av = Av_0$, we have the lost head expressed in terms of the velocity head in the pipe as

$$h' = \left(\frac{1}{c_0^2} - 1 \right) \left(\frac{a}{A} \right) \frac{v^2}{2g} \cdot \cdot \cdot \cdot \cdot \quad (6)$$

7. *Tables and Figures.*—Table 2 gives the condensed experi-

TABLE 2
CONDENSED EXPERIMENTAL DATA FOR 6-INCH SUBMERGED PIPE HAVING
CONICAL MOUTHPIECES
(Depth of Submergence, about $3\frac{1}{2}$ Diameters)

Mouth-piece Attached to Pipe		Head Causing Flow Feet	Measured Discharge Cubic Feet per Second	Mean Velocity in Pipe Feet per Second	Coefficient of Discharge Based on Area of Pipe	Mouth-piece Attached to Pipe		Head Causing Flow Feet	Measured Discharge Cubic Feet per Second	Mean Velocity in Pipe Feet per Second	Coefficient of Discharge Based on Area of Pipe
Entrance	Discharge					Entrance	Discharge				
		<i>h</i>	<i>q</i>	<i>v</i>	<i>c</i>			<i>h</i>	<i>q</i>	<i>v</i>	<i>c</i>
None (Inward-Projecting)	None	.0074	.102	.520	.756	90° Area Ratio (Flush Entrance) 1 to 3	None	.0075	.104	.530	.766
		.0128	.135	.689	.757			.0100	.123	.625	.778
		.0153	.153	.778	.779			.0114	.131	.665	.775
		.0190	.170	.868	.785			.0135	.144	.735	.783
		.0292	.210	1.07	.785			.0148	.151	.770	.793
		.0327	.223	1.14	.784			.0215	.185	.940	.799
		.0563	.294	1.50	.785			.0294	.224	1.14	.807
		.0745	.336	1.71	.783			.0310	.215	1.10	.795
		.1112	.405	2.07	.784			.0428	.257	1.32	.795
		.1976	.551	2.81	.787			.0535	.290	1.48	.798
		.296	.668	3.41	.780			.0942	.389	1.98	.802
		.4420	.819	4.17	.783			.1727	.520	2.65	.798
								.1840	.540	2.76	.802
								.2636	.645	3.28	.802
								.3410	.739	3.76	.803
5° Area Ratio 1 to 2	None	.0093	.130	.665	.854	None	5° Area Ratio 1 to 2	.0172	.198	1.01	.961
		.0157	.176	.899	.886			.0310	.271	1.38	.975
		.0357	.265	1.35	.895			.0313	.273	1.39	.976
		.0503	.319	1.63	.903			.0408	.311	1.58	.977
		.0750	.432	2.20	.908			.0489	.344	1.75	.986
		.0880	.425	2.16	.904			.0589	.378	1.92	.985
		.0886	.425	2.16	.906			.0700	.411	2.10	.987
		.1700	.577	2.94	.895			.0930	.473	2.41	.988
		.2234	.671	3.42	.903			.1060	.502	2.56	.980
		.3520	.838	4.27	.899			.1135	.524	2.67	.986
								.1155	.525	2.67	.978
								.1400	.594	3.03	1.01
								.1633	.623	3.17	.980
								.2560	.792	4.04	.994
								.3670	.948	4.83	.993
10° Area Ratio 1 to 2	None	.0089	.130	.661	.875	None	10° Area Ratio 1 to 2	.0182	.193	.985	.910
		.0087	.132	.673	.899			.0361	.278	1.42	.930
		.0116	.146	.743	.869			.0600	.362	1.85	.939
		.0170	.183	.934	.893			.0807	.418	2.13	.932
		.0309	.250	1.28	.903			.1335	.543	2.76	.944
		.0378	.276	1.41	.902			.1560	.589	3.00	.945
		.0545	.333	1.70	.906			.1680	.606	3.08	.938
		.0582	.340	1.73	.893			.2930	.790	4.02	.934
		.0600	.347	1.77	.900			.4120	.960	4.89	.943
		.0656	.369	1.88	.915						
		.0798	.400	2.04	.899						
		.1005	.453	2.31	.908						
		.1057	.465	2.37	.910						
		.1880	.621	3.16	.910						
		.2170	.672	3.42	.913						
		.2540	.730	3.72	.920						
		.2557	.722	3.68	.908						
		.3090	.797	4.06	.910						

TABLE 2 (Continued)
 CONDENSED EXPERIMENTAL DATA FOR 6-INCH SUBMERGED PIPE HAVING
 CONICAL MOUTHPIECES
 (Depth of Submergence, about $3\frac{1}{2}$ Diameters)

Mouth-piece Attached to Pipe		Head Causing Flow Feet	Measured Discharge Cubic Feet per Second	Mean Velocity in Pipe Feet per Second	Coefficient of Discharge Based on Area of Pipe	Mouth-piece Attached to Pipe		Head Causing Flow Feet	Measured Discharge Cubic Feet per Second	Mean Velocity in Pipe Feet per Second	Coefficient of Discharge Based on Area of Pipe
Entrance	Discharge					Entrance	Discharge				
		<i>h</i>	<i>q</i>	<i>v</i>	<i>c</i>			<i>h</i>	<i>q</i>	<i>v</i>	<i>c</i>
15° Area Ratio 1 to 2	None	.0161	.175	.893	.878	None	15° Area Ratio 1 to 2	.0120	.146	.743	.845
		.0288	.241	1.23	.902			.0185	.190	.968	.888
		.0450	.303	1.54	.906			.0230	.200	1.02	.861
		.0582	.350	1.78	.921			.0406	.281	1.43	.884
		.0633	.365	1.86	.909			.0444	.294	1.50	.885
		.0662	.405	2.07	.911			.0595	.344	1.75	.895
		.0852	.417	2.12	.906			.0610	.349	1.78	.900
		.1030	.463	2.36	.917			.0660	.367	1.87	.902
		.1827	.622	3.17	.926			.0670	.365	1.86	.904
		.2540	.728	3.71	.915			.0898	.428	2.18	.905
		.3520	.864	4.40	.925			.0891	.425	2.16	.901
		.4230	.952	4.85	.925			.1242	.500	2.55	.899
		.7650	1.25	6.40	.915			.1692	.578	2.95	.902
								.2637	.729	3.72	.893
								.2820	.746	3.80	.907
								.3400	.833	4.24	
20° Area Ratio 1 to 2	None	.0080	.123	.628	.877	None	20° Area Ratio 1 to 2	.0090	.123	.625	.821
		.0105	.147	.752	.914			.0148	.161	.821	.844
		.0191	.196	1.00	.905			.0153	.162	.827	.834
		.0306	.252	1.28	.912			.0160	.167	.849	.837
		.0355	.269	1.37	.908			.0244	.211	1.07	.859
		.0550	.338	1.72	.917			.0383	.269	1.37	.870
		.0695	.381	1.94	.917			.0404	.271	1.38	.856
		.0754	.394	2.01	.912			.0404	.272	1.38	.853
		.0846	.416	2.12	.908			.0437	.283	1.44	.860
		.1380	.540	2.75	.920			.0505	.305	1.55	.864
		.1736	.606	3.09	.924			.0747	.372	1.89	.863
		.2320	.707	3.60	.927			.0782	.385	1.96	.876
		.3075	.804	4.10	.921			.0953	.423	2.15	.870
		.4270	.964	4.91	.927			.1385	.514	2.62	.877
								.1867	.598	3.05	.879
								.2025	.626	3.19	.884
								.330	.795	4.05	.880
30° Area Ratio 1 to 2	None	.0080	.121	.619	.862	None	30° Area Ratio 1 to 2	.0096	.115	.586	.746
		.0083	.124	.633	.866			.0152	.146	.744	.751
		.0185	.187	.955	.874			.0152	.153	.779	.789
		.0221	.212	1.08	.907			.0176	.165	.841	.791
		.0367	.272	1.38	.900			.0291	.211	1.08	.786
		.0410	.287	1.46	.900			.0506	.281	1.43	.792
		.0470	.309	1.58	.906			.0519	.283	1.44	.789
		.0594	.347	1.77	.902			.0804	.359	1.83	.792
		.0804	.406	2.07	.911			.0852	.367	1.87	.798
		.0818	.414	2.11	.919			.1690	.513	2.61	.793
		.0967	.448	2.28	.915			.2331	.606	3.08	.797
		.1524	.563	2.87	.916			.2715	.657	3.35	.802
		.2404	.710	3.62	.921			.3990	.794	4.04	.798
		.3341	.838	4.27	.921						
		.390	.906	4.61	.923						
45° Area Ratio 1 to 2	None	.0189	.192	.975	.884	None	45° Area Ratio 1 to 2	.0078	.102	.521	.733
		.0278	.231	1.18	.885			.0163	.154	.787	.773
		.0404	.281	1.43	.888			.0340	.227	1.16	.785
		.0462	.304	1.55	.902			.0521	.282	1.44	.790
		.0721	.381	1.94	.902			.0546	.288	1.46	.782
		.0762	.386	1.97	.889			.0774	.343	1.75	.784
		.0965	.438	2.23	.896			.1185	.426	2.17	.785
		.1535	.557	2.84	.902			.2480	.617	3.14	.787
		.1857	.615	3.14	.907			.3970	.782	3.98	.789
		.325	.803	4.09	.890						
		.399	.902	4.59	.907						
		.451	.961	4.89	.910						

TABLE 2 (Continued)
 CONDENSED EXPERIMENTAL DATA FOR 6-INCH SUBMERGED PIPE HAVING
 CONICAL MOUTHPIECES
 (Depth of Submergence, about $3\frac{1}{2}$ Diameters)

Entrance	Discharge	Head Causing Flow Feet	Meas- ured Dis- charge Cubic Feet per Second	Mean Velocity in Pipe Feet per Second	Coeffi- cient of Dis- charge Based on Area of Pipe	Entrance	Discharge	Head Causing Flow Feet	Meas- ured Dis- charge Cubic Feet per Second	Mean Velocity in Pipe Feet per Second	Coeffi- cient of Dis- charge Based on Area of Pipe
		<i>h</i>	<i>q</i>	<i>v</i>	<i>c</i>			<i>h</i>	<i>q</i>	<i>v</i>	<i>c</i>
60° Area Ratio 1 to 2	None	.0079	.119	.606	.852	None	60° Area Ratio 1 to 2	.0116	.126	.644	.743
		.0083	.120	.614	.845			.0154	.149	.758	.762
		.0159	.174	.886	.876			.0396	.243	1.24	.776
		.0255	.219	1.11	.870			.0400	.246	1.25	.780
		.0383	.272	1.38	.883			.0577	.296	1.51	.780
		.0526	.316	1.61	.874			.0600	.304	1.55	.786
		.0553	.326	1.66	.877			.1035	.399	2.03	.787
		.0798	.394	2.01	.889			.1786	.526	2.68	.790
		.0800	.391	1.99	.880			.1940	.547	2.79	.792
		.1088	.465	2.37	.895			.2605	.642	3.27	.796
		.1437	.526	2.68	.882			.3101	.696	3.55	.795
		.2657	.716	3.65	.883			.4777	.859	4.38	.790
		.4129	.896	4.56	.886						
10° Area Ratio 1 to 3	None	.0080	.128	.654	.907	None	10° Area Ratio 1 to 3	.0103	.147	.750	.920
		.0150	.177	.900	.900			.0135	.168	.858	.924
		.0160	.179	.911	.899			.0185	.201	1.03	.941
		.0170	.185	.942	.901			.0304	.255	1.30	.931
		.0181	.190	.968	.897			.0319	.264	1.35	.940
		.0187	.196	1.00	.913			.0461	.317	1.61	.938
		.0198	.200	1.02	.902			.0530	.345	1.76	.956
		.0210	.210	1.07	.922			.0600	.360	1.83	.929
		.0500	.325	1.66	.922			.0749	.403	2.06	.934
		.0505	.325	1.66	.918			.0750	.404	2.06	.938
		.0516	.331	1.69	.924			.1000	.469	2.39	.951
		.0770	.398	2.03	.914			.1028	.479	2.44	.948
		.0960	.452	2.30	.928			.1378	.548	2.80	.938
		.1176	.497	2.53	.921			.1450	.572	2.92	.954
		.1570	.574	2.92	.921			.1740	.628	3.20	.956
		.1737	.609	3.11	.929			.1880	.662	3.37	.967
		.2375	.708	3.61	.925			.2175	.699	3.56	.954
		.2585	.745	3.80	.930			.2751	.794	4.04	.961
15° Area Ratio 1 to 3	None	.0080	.129	.660	.916	None	15° Area Ratio 1 to 3	.0090	.133	.675	.885
		.0165	.187	.955	.925			.0145	.170	.866	.896
		.0332	.257	1.31	.930			.0149	.169	.862	.880
		.0497	.324	1.65	.921			.0152	.172	.874	.880
		.0513	.330	1.68	.926			.0293	.242	1.24	.897
		.0778	.400	2.04	.912			.0588	.340	1.74	.893
		.1058	.476	2.34	.930			.0589	.345	1.76	.901
		.1200	.505	2.57	.927			.0812	.401	2.04	.896
		.1924	.638	3.25	.924			.1060	.466	2.38	.910
		.3634	.880	4.49	.928			.1238	.501	2.55	.905
								.1855	.610	3.11	.901
								.2043	.648	3.30	.914
								.2380	.705	3.59	.918
								.2685	.742	3.78	.910
								.2895	.777	3.96	.918
20° Area Ratio 1 to 3	None	.0080	.126	.644	.895	None	20° Area Ratio 1 to 3	.0080	.119	.607	.845
		.0140	.172	.874	.921			.0141	.160	.818	.867
		.0260	.234	1.19	.919			.0185	.182	.925	.848
		.0630	.366	1.87	.925			.0224	.204	1.04	.864
		.1010	.463	2.36	.924			.0225	.215	1.10	.855
		.1095	.483	2.46	.927			.0414	.278	1.41	.867
		.2101	.687	3.50	.930			.0440	.284	1.45	.872
		.3614	.880	4.49	.929			.0503	.310	1.58	.879
								.0855	.411	2.10	.892
								.1199	.486	2.48	.890
								.2029	.633	3.23	.892
								.2422	.694	3.54	.892
								.3071	.782	3.99	.897

TABLE 2 (Continued)

CONDENSED EXPERIMENTAL DATA FOR 6-INCH SUBMERGED PIPE HAVING
CONICAL MOUTHPIECES
(Depth of Submergence, about $3\frac{1}{2}$ Diameters)

Mouth- piece Attached to Pipe		Head Causing Flow Feet	Meas- ured Dis- charge Cubic Feet per Second	Mean Velocity in Pipe Feet per Second	Coeff- icient of Dis- charge Based on Area of Pipe	Mouth- piece Attached to Pipe		Head Causing Flow Feet	Meas- ured Dis- charge Cubic Feet per Second	Mean Velocity in Pipe Feet per Second	Coeff- icient of Dis- charge Based on Area of Pipe
Entrance	Discharge					Entrance	Discharge				
30° Area Ratio 1 to 4	None	.0097	.144	.736	.932	None	30° Area Ratio 1 to 4	.0085	.122	.623	.841
		.0106	.149	.761	.920			.0206	.192	.980	.851
		.0306	.254	1.29	.922			.0241	.214	1.09	.874
		.0440	.306	1.56	.925			.0475	.297	1.51	.865
		.0599	.353	1.80	.918			.0584	.335	1.71	.880
		.0755	.397	2.02	.920			.0803	.394	2.03	.884
		.1079	.475	2.42	.920			.1112	.463	2.36	.882
		.1525	.570	2.91	.928			.1276	.500	2.55	.889
		.1870	.632	3.22	.930			.1600	.566	2.88	.899
		.1906	.638	3.25	.927			.2203	.666	3.39	.902
		.2801	.780	3.97	.936			.2789	.756	3.85	.910
								.2975	.777	3.98	.905
30° Area Ratio 1 to 3	None	.0085	.124	.630	.853	None	30° Area Ratio 1 to 3	.0076	.107	.546	.780
		.0165	.178	.909	.881			.0082	.106	.543	.770
		.0235	.220	1.12	.909			.0147	.150	.764	.785
		.0301	.247	1.26	.906			.0195	.173	.883	.788
		.0303	.247	1.26	.901			.0300	.215	1.09	.787
		.0643	.369	1.88	.925			.0310	.221	1.12	.796
		.0663	.374	1.91	.923			.0320	.217	1.11	.794
		.0950	.450	2.29	.928			.0520	.281	1.43	.795
		.1730	.611	3.11	.933			.0831	.360	1.83	.794
		.3053	.810	4.13	.932			.1454	.480	2.44	.800
								.2901	.682	3.47	.803
								.4600	.855	4.36	.801
45° Area Ratio 1 to 3	None	.0080	.119	.607	.845	None	45° Area Ratio 1 to 3	.0115	.131	.666	.773
		.0129	.153	.780	.853			.0178	.162	.825	.771
		.0232	.214	1.09	.893			.0305	.215	1.09	.779
		.0262	.231	1.17	.905			.0580	.297	1.52	.784
		.0412	.285	1.45	.890			.0993	.390	1.98	.785
		.0602	.346	1.76	.895			.1493	.475	2.42	.781
		.0783	.397	2.02	.901			.1970	.546	2.79	.782
		.1164	.490	2.50	.912			.2896	.669	3.41	.789
		.1256	.502	2.56	.900			.350	.737	3.76	.795
		.1801	.600	3.06	.898			.452	.836	4.26	.792
		.2233	.668	3.40	.896			.507	.917	4.67	.796
		.3117	.788	4.02	.897						
		.3223	.813	4.14	.906						
20° Area Ratio 1 to 2	5° Area Ratio 1 to 2	.0130	.232	1.18	1.29	20° Area Ratio 1 to 2	5° Area Ratio 1 to 2	.0091	.190	.968	1.27
		.0209	.301	1.54	1.32			.0221	.302	1.54	1.29
		.0280	.348	1.77	1.32			.0413	.416	2.12	1.30
		.0504	.464	2.37	1.32			.0626	.519	2.65	1.32
		.0520	.484	2.46	1.34			.1900	.686	3.49	1.35
		.0651	.539	2.75	1.34						
		.1083	.700	3.56	1.35						
		.1548	.836	4.26	1.35						
20° Area Ratio 1 to 2	10° Area Ratio 1 to 2	.0076	.158	.802	1.15	20° Area Ratio 1 to 2	10° Area Ratio 1 to 2	.0194	.261	1.33	1.19
		.0204	.263	1.34	1.17			.0274	.317	1.61	1.22
		.0397	.371	1.89	1.18			.0480	.345	2.16	1.23
		.0628	.470	2.35	1.18			.0734	.519	2.64	1.21
		.0915	.476	2.89	1.19			.1110	.639	3.26	1.22
		.1449	.708	3.60	1.18						
		.496	1.33	6.80	1.18						

TABLE 2 (Concluded)
CONDENSED EXPERIMENTAL DATA FOR 6-INCH SUBMERGED PIPE HAVING
CONICAL MOUTHPIECES
(Depth of Submergence, about $3\frac{1}{2}$ Diameters)

Mouth-piece Attached to Pipe		Head Causing Flow Feet	Measured Discharge Cubic Feet per Second	Mean Velocity in Pipe Feet per Second	Coefficient of Discharge Based on Area of Pipe	Mouth-piece Attached to Pipe		Head Causing Flow Feet	Measured Discharge Cubic Feet per Second	Mean Velocity in Pipe Feet per Second	Coefficient of Discharge Based on Area of Pipe
Entrance	Discharge					Entrance	Discharge				
		<i>h</i>	<i>q</i>	<i>v</i>	<i>c</i>			<i>h</i>	<i>q</i>	<i>v</i>	<i>c</i>
15° Area Ratio 1 to 2	15° Area Ratio 1 to 2	.0062	.128	.650	1.03	20° Area Ratio 1 to 2	15° Area Ratio 1 to 3	.0089	.160	.817	1.08
		.0192	.239	1.22	1.09			.0098	.170	.866	1.10
		.0433	.363	1.85	1.11			.0118	.196	1.00	1.14
		.0643	.443	2.25	1.11			.0224	.261	1.33	1.11
		.0690	.452	2.30	1.09			.0260	.292	1.49	1.15
		.3810	.975	4.97	1.09			.0456	.377	1.92	1.12
20° Area Ratio 1 to 3	20° Area Ratio 1 to 2	.0152	.191	.971	.984	20° Area Ratio 1 to 2	20° Area Ratio 1 to 3	.0388	.312	1.59	1.00
		.0333	.283	1.44	.985			.0651	.410	2.09	1.02
		.0621	.407	2.48	1.04			.0907	.494	2.52	1.04
		.0898	.486	2.07	1.03			.1434	.621	3.16	1.04
		.1512	.643	3.27	1.06			.1791	.695	3.55	1.04
		.233	.804	4.09							
20° Area Ratio 1 to 2	20° Area Ratio 1 to 4	.0343	.319	1.63	1.09	30° Area Ratio 1 to 2	10° Area Ratio 1 to 2	.0077	.160	.817	1.16
		.0559	.400	2.04	1.08			.0237	.282	1.44	1.17
		.0848	.481	2.46	1.05			.0340	.334	1.70	1.13
		.1233	.605	3.08	1.10			.0437	.381	1.94	1.16
		.1750	.726	3.70	1.10			.0630	.464	2.36	1.17
								.0650	.464	2.36	1.16
30° Area Ratio 1 to 2	30° Area Ratio 1 to 2	.0227	.219	1.11	.921	45° Area Ratio 1 to 2	45° Area Ratio 1 to 2	.0108	.141	.721	.867
		.0625	.363	1.85	.922			.0301	.246	1.25	.901
		.0671	.381	1.94	.932			.0525	.330	1.68	.912
		.0719	.396	2.02	.940			.0633	.357	1.82	.900
		.1009	.469	2.37	.932			.0966	.443	2.25	.904
		.1560	.585	2.98	.944			.1411	.535	2.73	.905
30° Area Ratio 1 to 2	30° Area Ratio 1 to 2	.1693	.608	3.11	.940	45° Area Ratio 1 to 2	45° Area Ratio 1 to 2	.2254	.681	3.47	.911
		.293	.796	4.06	.934			.466	.985	5.02	.912
		.373	.903	4.60	.932			.828	1.32	6.71	.915
		.465	1.07	5.49	.935						
		.742	1.27	6.48	.935						

TABLE 3
RESULTS OF EARLIER EXPERIMENTS ON CONVERGING MOUTHPIECES¹

Source	Coefficient of Discharge Based on Smallest Area	Mean Velocity at Smallest Area Feet per Second	Diameter or Side of Smallest Area	Ratio of Length of Straight Pipe Attached to d	Form of Mouthpiece
	c	v	d	$\frac{1}{d}$	
Balch	.73 to .92	4 to 15	4 in diam.	0.25 to 1.5	Conical, 45 degrees. Area Ratio 1 to 1.9
Davis and Balch	.829 to .955	4.5 to 13	1.51 ft. sq.	0.65	Approach to each side of square was circular. Area Ratio 1 to 3. Discharge vertical
Stewart	.928 to .894	1 to 4	4 ft. sq.	.077 to 3.5	Approach to each side of square was a quarter ellipse, the outer area being 8 feet square, the plane of which was 3 feet from entrance to straight pipe. Area Ratio 1 to 4
Ellis	.951 to .944	12 to 32	1 ft. sq.	0	Approach to each side of square was a quarter ellipse with semi-diameters of 0.5 feet (vertical) and 0.33 feet (horizontal). Discharge vertical
Francis	.927 to .944	5.5 to 9.2	1.22 in diam.	.082	Cycloidal. Area Ratio 1 to 1.95
Brownlee	.941 to .966	7.5 to 27.2	0.1982 in. diam.	0	Cycloidal. Area Ratio 1 to 4
Weisbach	.959 to .994	?	0.396 in diam.	0	Same as form of jet issuing from sharp edged orifice. Discharge into air
Castel	Angles		0.6 in diam.	Area Ratio	Conical. c varied but little with v . "Angle" means one-half total angle of convergence. $1/d = \text{zero}$. There was contraction at exit. Discharge into air
	Degrees	Minutes			
	.924 } .946 } .924 } .895 } .870 } .930 } .956 } .920 }	5 } 13 } 19 } 30 } 40 } 5 } 13 } 35 }			
		26 } 24 } 24 } 00 } 20 } 26 } 40 } 52 }			
			0.782 in diam.	$\left\{ \begin{array}{l} 1 \text{ to } 1.5 \\ \text{to} \\ 1 \text{ to } 1.88 \\ 1 \text{ to } 1.7 \\ \text{to} \\ 1 \text{ to } 10 \end{array} \right.$	

¹ For reference to source see Appendix.

mental data in which each set of values represents, as a rule, the average of three experiments or runs although in some cases as many as six or eight experiments were made under practically the same head as explained above. In some cases, also, experiments were repeated at times differing by several days or even weeks. This was done mainly to check the work of different experimenters and to determine the effect of certain factors which caused trouble during the experimenting.

In Figs. 6, 7, 8, and 9 the experimental values of the coefficients of discharge for the short pipe with the various mouthpieces attached

TABLE 4

RESULTS OF EARLIER EXPERIMENTS IN WHICH DISCHARGE MOUTHPIECES WERE USED¹

Source	Form and Angle of Entrance Mouthpiece or Approach	Area Ratio	Angle of Discharge Mouthpiece (One-Half of Total Angle)		Area Ratio	Coefficient of Discharge Based on Smallest Area	Diameter or Side of Smallest Section Inches	Connection
			Degrees	Minutes				
Weisbach	Sharp edge (no mouthpiece)		2	27	$\frac{1}{1.72}$	0.946	0.972	
Eytelwein	Circular	45 $\frac{1}{2}$	2	35	$\frac{1}{3}$	1.55	1.06	Connected without a straight pipe or throat
Francis	Cycloidal	45 $\frac{1}{1.95}$	2	30	$\frac{1}{2}$	1.55	1.22	Connected by a straight pipe or throat 0.1 inch long
Francis	Cycloidal	45 $\frac{1}{1.95}$	2	30	$\frac{1}{5.3}$	2.14	1.22	
Francis	Cycloidal	45 $\frac{1}{1.95}$	2	30	$\frac{1}{10}$	2.35	1.22	
Francis	Cycloidal	45 $\frac{1}{1.95}$	2	30	$\frac{1}{16}$	2.35	1.22	
Brownlee	Cycloidal	30 $\frac{1}{4}$	{ 3 33 } { slightly curved }		$\frac{1}{22}$	2.20	0.1982	Connected without a straight pipe
Venturi	Rounded corners	?	2	43	$\frac{1}{2.7}$	1.46	1.33	Connected without a straight pipe
Davis and Balch	Circular	45 $\frac{1}{3}$	14 +		$\frac{1}{1.78}$	1.03	12.51 (sq.)	Connected by a straight pipe 6 inches long. Vertical discharge
	Circular	45 $\frac{1}{3}$	curved		$\frac{1}{1.78}$	1.00	12.51 (sq.)	

¹ For references to source see Appendix. Velocities range about as in Table 3.² Approximate.

have been plotted as ordinates, and the mean velocities in the pipe as abscissas. It will be noted that the mean velocity of flow through the pipe varies from about 0.5 ft. to 6 ft. per sec., although in many cases the upper limit was between 4 and 5 ft. per sec. These curves show that in all cases the coefficient of discharge is practically constant for velocities above 2 ft. per sec. and in some cases above 1 ft. per sec. However, for discharge mouthpieces having small angles of divergence the curves show a tendency for the coefficient of discharge to increase slightly with the velocity. This, it is thought, is due chiefly to the more turbulent condition of the water in the downstream compartment caused by these longer mouthpieces, which in turn made the measurement of the water level slightly too large. For the same reason the results for the longer discharge mouthpieces show the greater fluctuations. In all

TABLE 5

FINAL RESULTS FOR 6-INCH SUBMERGED SHORT PIPE HAVING MOUTHPIECES WITH AN AREA RATIO
OF 1 TO 2 FOR VELOCITIES FROM 2 TO 5 FEET PER SECOND
(Depth of Submergence about $3\frac{1}{2}$ Diameters)

Mouthpiece Attached to Short Pipe		Area Ratio 1 to 2		Coefficient of Discharge from Experiment Based on Area of Pipe		Coefficient of Loss Based on Velocity Head in Pipe		Coefficient of Discharge Based on Outer Area of Mouthpiece		Coefficient of Loss Based on Outer Area of Mouthpiece $\left(\frac{1}{c_o^2} - 1\right)$		Head Required on Pipe with No Mouthpiece Attached to Give Same Discharge $H = \left(\frac{c}{.785}\right)^2 h$		Gain in Equivalent Head for Discharge Mouthpiece Due to Entrance Mouthpiece (See col. 8, table 7)		Coefficient of Discharge for Discharge Mouthpiece when Entrance Mouthpiece is Attached $c_d = .785 \times \sqrt{\text{Col. 7} + \text{Col. 8}}$		Per Cent Increase in c for Discharge Mouthpiece Due to Entrance Mouthpiece		Per Cent Gain in Velocity Head Recovered by Discharge Mouthpiece Due to Entrance Mouthpiece, Expressed in Per Cent of Maximum Theoretical Amount Possible of Recovery. (See cols. 11 and 12, table 7)	
Entrance Degrees	Discharge Degrees	c	m			$\left(\frac{1}{c^2} - 1\right)$ Column 6 times $\left(\frac{a}{A}\right)^2$	c_o														
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22
0	0	.900 .785	.235 .620	.495	3.08	1.32h	.51h	1.137	14.9	33.0											
5	5	.990	.770	.495	3.08	1.34h	.51h	1.137	14.9	33.0											
10	10	.910	.940	.470	3.50	1.38h	.19h	1.000	6.39	19.0											
15	15	.920	.975	.450	3.90	1.38h	.11h	.938	4.22	7.9											
20	20	.925	.880	.440	4.15	1.39h	.01h	.884	0.45	0											
30	30	.920	.800	.400	5.25	1.38h	0	.800	0	0											
45	45	.900	.790	.395	5.40	1.32h	0	.790	0	0											
60	60	.885	.790	.395	5.40	1.27h	0	.790	0	0											
..	60	.790	1.37	.395	5.40	1.01h	0	.790	0	0											

1 Angle given is one-half total angle of convergence or divergence

² This is called "equivalent head."

TABLE 6
FINAL RESULTS FOR A 6-INCH SUBMERGED SHORT PIPE HAVING MOUTHPIECES WITH AN AREA RATIO
OF 1 TO 3 FOR VELOCITIES FROM 2 TO 5 FEET PER SECOND
(Depth of Submergence about $3\frac{1}{2}$ Diameters)

Entrance Degrees	Discharge Degrees	Coefficient of Discharge from Experiment Based on Area of Pipe	Coefficient of Loss Based on Velocity Head in Pipe		Coefficient of Discharge Based on Outer Area of Mouthpiece	Coefficient of Loss Based on Velocity Head at Outer Area of Mouthpiece $\left(\frac{1}{c_o^2} - 1\right)$	Head Required on Pipe with No Mouthpiece Attached to Give Same Discharge $H = \left(\frac{c}{.785}\right)^2 h$		Gain in Equivalent Head for Discharge Mouthpiece Due to Entrance Mouthpiece (See col. 8, table 7)	Coefficient of Discharge for Discharge Mouthpiece when Entrance Mouth- piece is Attached $c_d = .785 \times \sqrt{\text{Col. 7} + \text{Col. 8}}$	Per Cent Increase in c for Discharge Mouthpiece Due to Entrance Mouthpiece	Per Cent Increase in Velocity Head Recovered by Discharge Mouthpiece Due to En- trance Mouthpiece, Ex- pressed in Per Cent of Maximum Theoretical Amount Possible of Re- covery. (See cols. 11 and 12, table 7)
			$\left(\frac{1}{c^2} - 1\right)$ Column 6 times $\left(\frac{a}{A}\right)^2$	m			H_E	H_D				
1	2	3	4	5	6	7	8	9	10	11		
0	0	.925 .785	.165 .620	.317	9.00	1.39 <i>h</i>	0.27 <i>h</i>	1.035	8.95	14.7		
10	10	.925	.165	.303	9.86	1.39 <i>h</i>	0.17 <i>h</i>	.958	5.28	11.5		
15	15	.910	1.095	.297	10.30	1.32 <i>h</i>		.890	0	0		
20	20	.925	.165	.267	13.00	1.28 <i>h</i>	0	.800	0	0		
30	30	.890	1.14	.263	13.45	1.39 <i>h</i>	0	.790	0	0		
45	45	.800	1.44			1.04 <i>h</i>						
60	60	.790	1.49			1.01 <i>h</i>						
90	90	.800	.560			1.04 <i>h</i>						
20	20	.925	.165			1.39 <i>h</i>						
(1 to 4)	(1 to 4)	.900	1.23	.225	18.70	1.31 <i>h</i>	0.04 <i>h</i>	.912	1.33	9.0		

1 Angle given is one-half total angle of convergence or divergence.

TABLE 7
FINAL RESULTS FOR 6-INCH SUBMERGED SHORT PIPE HAVING A MOUTHPIECE ATTACHED TO EACH END FOR
VELOCITIES FROM 2 TO 5 FEET PER SECOND
(Depth of Submergence about $3\frac{1}{2}$ Diameters)

Mouthpiece Attached to Short Pipe	Area	Ratio	Coefficient of Discharge from Experiment Based on Area of Pipe		Same as Col. 3 Except Based on Outer Area of Mouthpiece		Coefficient of Loss Based on Velocity Head in Pipe $m_o = \left(\frac{1}{c_o^2} - 1\right) \left(\frac{a}{A}\right)^2$	"Equivalent Head" or Head on Straight Pipe to Give Same Discharge $H_S = \left(\frac{c}{.785}\right)^2 h$	That Part of Equivalent Head Due to Discharge Mouthpiece $H'_D = \frac{H_S}{H_E}$	Increase in Equivalent Head for Discharge Mouthpiece Due to Entrance Mouthpiece $H'_D - H_D$	Coefficient of Discharge Calculated from Data for Mouthpieces When Tested Separately $c' = .785 \times \sqrt{H_E \times H_D}$	Coefficient of Loss Based on c' but on Velocity Head in Pipe $m_o = \left(\frac{1}{c_o'^2} - 1\right) \left(\frac{a}{A}\right)^2$	Ratio of Velocity Head Recovered to Maximum Possible of Recovery (in Per Cent)	
			c		c_o									
Entrance Degrees	Discharge Degrees												Contraction Suppressed	Contraction not Suppressed
1	2	3	4	5	6	7	8	9	10	11	12			
20 (1-2)	5 (1-2)	1.34	.720	.232	2.92h	2.10h	0.51h	1.17	.480	91.0	58.0			
20 (1-2)	10 (1-3)	1.22	.407	.558	2.42h	1.74h	0.27h	1.12	.687	55.4	40.7			
20 (1-2)	10 (1-2)	1.18	.590	.468	2.26h	1.63h	0.19h	1.11	.610	59.7	40.7			
15 (1-2)	15 (1-2)	1.10	.550	.572	1.97h	1.43h	0.11h	1.06	.632	47.7	39.8			
20 (1-2)	20 (1-2)	1.04+	.347	.811	1.76h	1.28h	0.00h	1.04	.811	26.6	26.6			
20 (1-3)	20 (1-3)	1.04	.520	.672	1.76h	1.27h	0.01h	1.04	.672	32.4	32.4			
30 (1-2)	30 (1-2)	0.940	.470	.875	1.44h	1.04h	0.00h	0.940	.875	7.35	7.35			
45 (1-2)	45 (1-2)	0.910	.455	.955	1.34h	1.02h	0.01h	0.906	.962	4.00	4.00			
20 (1-2)	20 (1-2)	1.075	.270	.795	1.88h	1.35h	0.04h	1.065	.880	33.0	24.0			
20 (1-2)	15 (1-4)	1.13	.377	.675	2.07h	1.49h	0.17h	1.06	.777	42.0	30.5			

1 Angle given is one-half total angle of convergence or divergence.

cases, the coefficient of discharge decreases more or less rapidly as the velocity decreases from 1 or 2 ft. per sec.

Tables 5, 6, and 7 give the values of the coefficients of discharge and the corresponding values of the coefficients of loss for the short pipe with the various mouthpieces for velocities of from 2 to 5 ft. per sec. as taken from the curves in Figs. 6, 7, 8, and 9, together with certain other data used in the discussion which follows. Figs. 12 and 14 show the influence upon the action of discharge mouthpieces of attaching an entrance mouthpiece to the short pipe. The entrance mouthpiece suppresses the contraction at entrance to the pipe and allows the discharge mouthpiece to receive the water in a smoother state of flow than when the entrance end of the pipe is simply inward projecting. The influence of smooth flow is shown in Figs. 12 and 14 which are obtained by plotting the gain in rate of discharge and gain in velocity head recovered (expressed in per cent) as ordinates and the angle of discharge mouthpieces as abscissas.

8. *Inward-Projecting and Flush Entrance.*—The values of the coefficients of discharge for the short pipe with inward-projecting entrance (no mouthpiece attached) were determined with special care since the effect of attaching a mouthpiece could not otherwise be found. It will be noted from Fig. 6 and Tables 5 and 6 that the value of the coefficient of discharge and the coefficient of loss are respectively $c = 0.785$ and $m = 0.62$ while the values generally given in texts for an inward-projecting pipe are $c = 0.72$ and $m = 0.93$. That is, the head lost at the entrance to an inward projecting pipe is 0.62 of the velocity head in the pipe ($0.62 \frac{v^2}{2g}$) instead of $0.93 \frac{v^2}{2g}$.

It would hardly be expected that all inward-projecting pipes would give the same coefficient of discharge, for such factors as the condition of the edge at entrance to the pipe, the diameter of the pipe or perhaps the ratio of the thickness of the pipe to the diameter, the degree of wetness of the material (effect of oil, etc.), the temperature and velocity of the water, the form and size of tank together with the location and form of piezometer orifice, and the conditions of discharge (submerged or into air) might easily influence the flow.

In order to get further data on this form of entrance another short pipe, 3.11 in. in diameter by 12 in. long, was tested in the same

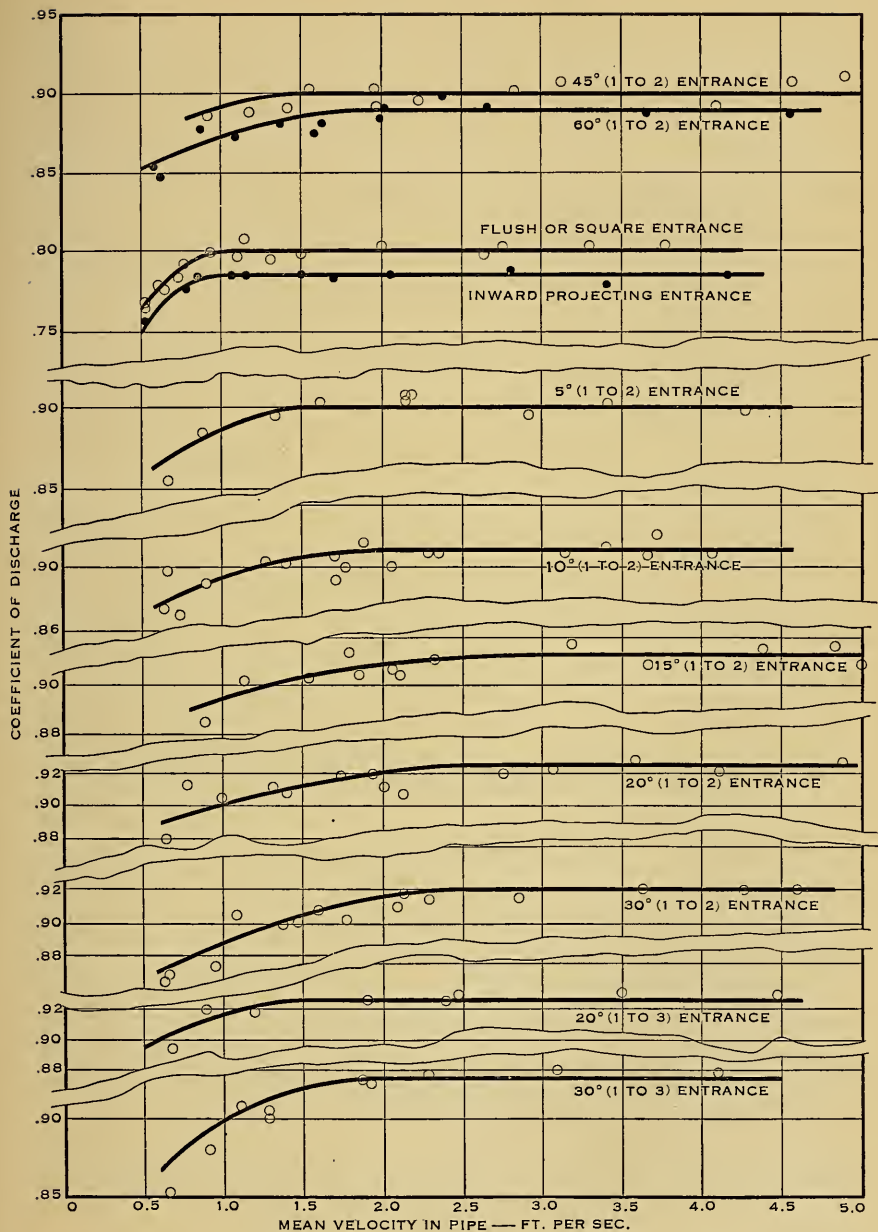


FIG. 6. RELATION BETWEEN COEFFICIENT OF DISCHARGE FOR THE SHORT PIPE WITH MOUTHPIECES AND THE MEAN VELOCITY IN PIPE

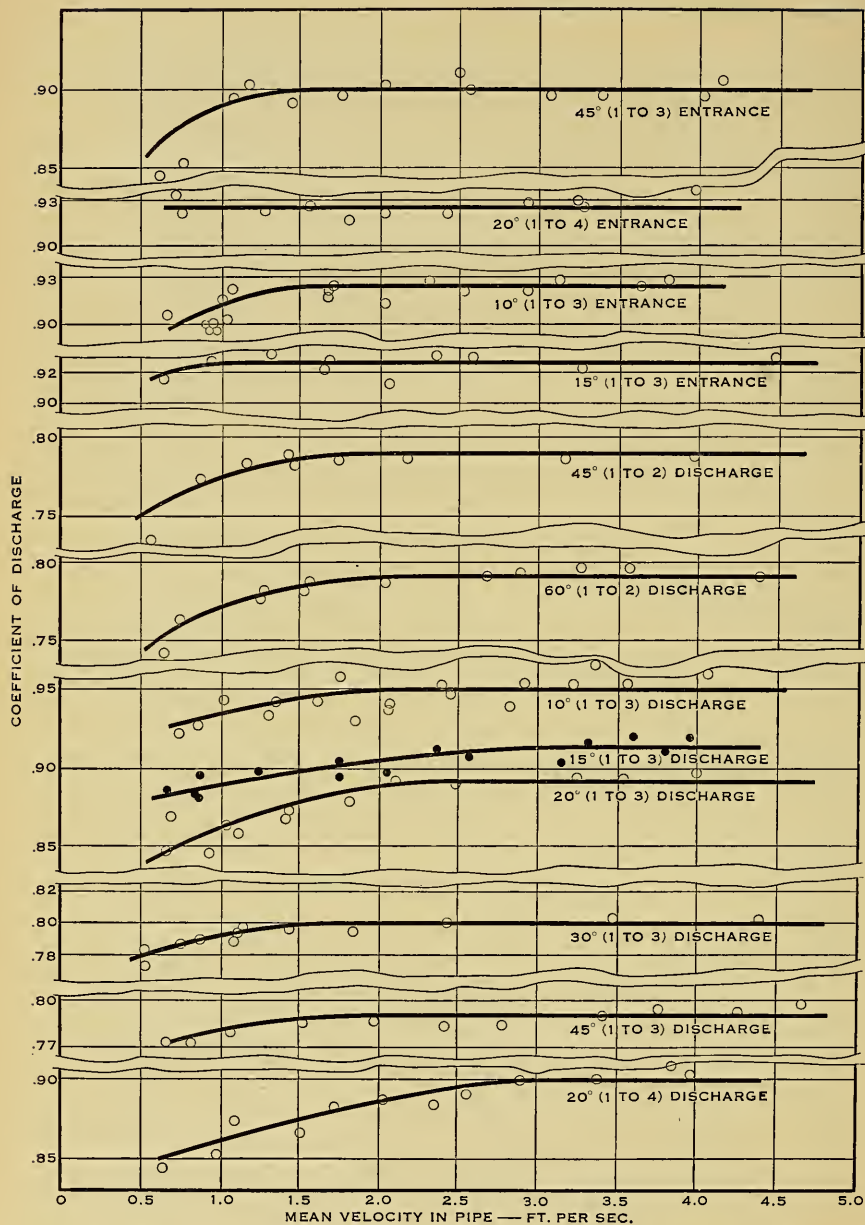


FIG. 7. RELATION BETWEEN COEFFICIENT OF DISCHARGE FOR THE SHORT PIPE WITH MOUTHPIECES AND THE MEAN VELOCITY IN PIPE (CONTINUED)

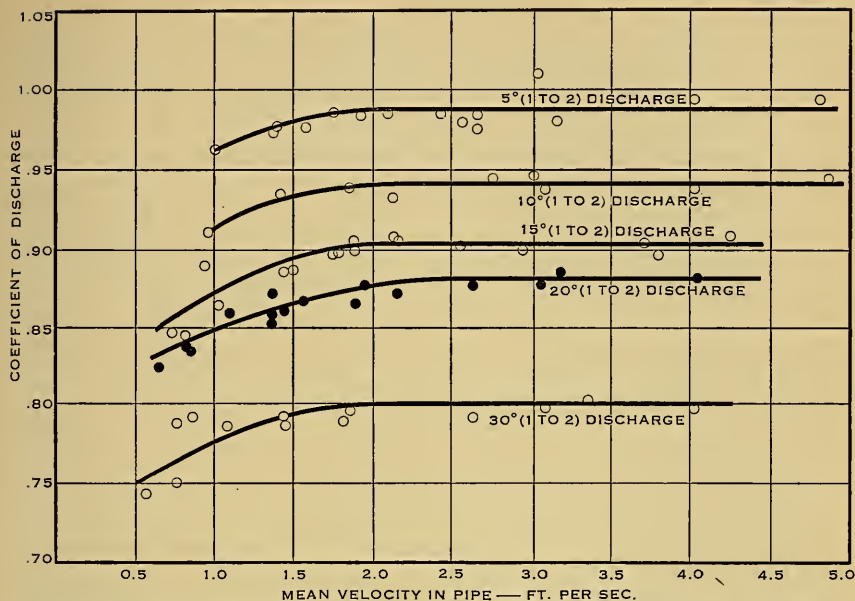


FIG. 8. RELATION BETWEEN COEFFICIENT OF DISCHARGE FOR THE SHORT PIPE WITH MOUTHPIECES AND THE MEAN VELOCITY IN PIPE (CONTINUED)

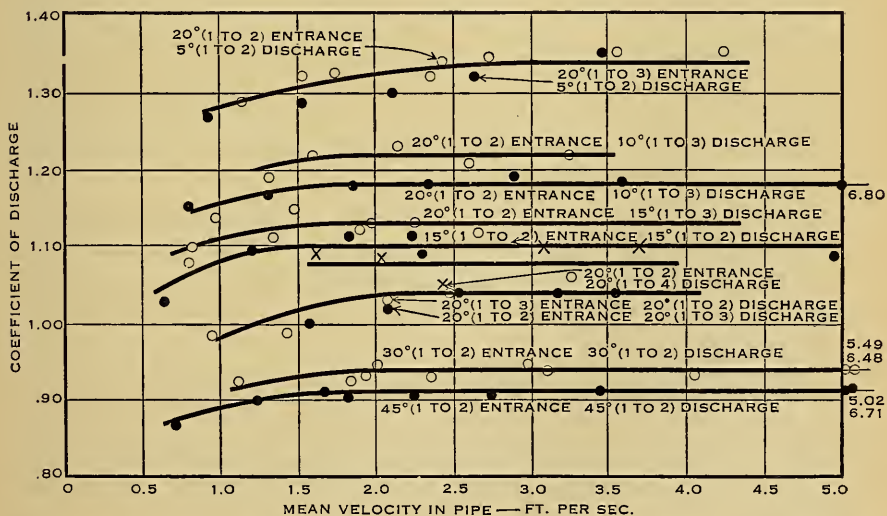


FIG. 9. RELATION BETWEEN COEFFICIENT OF DISCHARGE FOR THE SHORT PIPE WITH MOUTHPIECES AND THE MEAN VELOCITY IN PIPE (CONTINUED)

tank, the maximum velocity being slightly greater than 5 ft. per sec. The values of the coefficient of discharge varied but little, ranging from 0.783 to 0.788. The ratio of the thickness of the pipe at entrance to the diameter for the 6-in. pipe was 0.05 and for the 3-in. pipe this ratio was 0.054. The entering edge of the 3-in. pipe was somewhat sharper than that of the 6-in. pipe.

The value for the lost head at entrance to an inward-projecting pipe as usually given seems to be based on rather meager data obtained chiefly from experiments with discharge into air at rather high velocities through tubes of small diameters. The value $c = 0.72$ for the coefficient of discharge seems to have been handed down from Weisbach. Bidone* reported a value of $c = 0.767$ and Bilton* a value of $c = 0.75$ for a $2\frac{1}{2}$ -in. pipe, increasing to 0.79 for a 1-in. pipe and to 0.93 for a $\frac{1}{8}$ -in. pipe, the length in each case being $2\frac{1}{2}$ diameters.

The values of the coefficient of discharge and the coefficient of loss for a flush entrance (corresponding in these experiments to a 90-degree entrance mouthpiece having an area ratio of 1 to 3) also fail to check closely the values so generally given in texts and so generally used, namely, $c = 0.82$ and $m = 0.49$. As shown in Fig. 6 and Table 6, the values found in these experiments are $c = 0.80$ and $m = 0.56$. That is, the lost head at entrance to a short pipe having a flush entrance is $0.56 \frac{v^2}{2g}$. A 90-degree mouthpiece with an area

ratio of 1 to 3 is ample to give the same conditions of flow as a reservoir wall which is flush with end of the pipe and hence it should give the same rate of discharge.

More experimental data on short pipes having flush entrances are available, and they cover a wider range of sizes and conditions than do those for inward-projecting pipes, the value of the coefficient of discharge ranging from 0.785 to 0.84 with the majority of the values lying below 0.82.*

The lost head at the entrance to a pipe can probably be determined better from a submerged tube than from one discharging into air. It would seem, therefore, that the value of the lost head at the entrance to an inward-projecting pipe is not so different from that

*See Appendix for references.

for a pipe having a flush entrance as is usually believed, and that the common text book values need revision to apply to conditions commonly met in hydraulics.

9. *Entrance Mouthpieces.*—From Tables 5 and 6 and the curves in Figs. 7 and 8, it will be seen that entrance mouthpieces having angles of from 10 degrees to 30 degrees (20 degrees to 60 degrees total angle of convergence) give practically the same discharge, while all the entrance mouthpieces having angles of from 5 degrees to 60 degrees (10 degrees to 120 degrees total angle) give only about 5 per cent range in the rate of discharge. In other words, the lost head at the entrance to an inward-projecting short pipe may be reduced from 0.62 of the velocity head in the pipe to 0.18 of the velocity head by a conical mouthpiece having an angle ranging from 10 degrees to 30 degrees. Also, the lost head will vary but little from 0.20 of the velocity head in the pipe for all entrance mouthpieces having angles between 10 degrees and 45 degrees (20 degrees and 90 degrees total angle). It should be stated, however, that conditions surrounding the entrance to the mouthpiece may have some effect, such as the accumulation of dirt in a passage or any other obstruction. Furthermore, it is clear from a comparison of Figs. 7 and 8 that no advantage results from increasing the length of the entrance mouthpiece beyond that corresponding to an area ratio of 1 to 2. The lost head at the entrance to a mouthpiece is sometimes considered the same as would occur at the entrance to an inward projecting pipe of the same area as that of the mouthpiece; that is, the lost head is found by multiplying the velocity head at entrance to the mouthpiece by the coefficient of loss for an inward projecting pipe. There seems to be little reason to justify such a method, since an entrance mouthpiece having an area ratio of 1 to 3 gives almost the same lost head as one with an area ratio of 1 to 2 while the velocity head at entrance to the former mouthpiece would be, of course, only one ninth of that of the latter.

It is not clear just what effect a straight throat or pipe has when added to an entrance mouthpiece. The discharge through the mouthpieces alone was not determined in these experiments. It would seem that if the mouthpiece suppressed the contraction very completely, the pipe would cause added resistance only, and hence decrease the discharge, while if the suppression was rather incomplete, the pipe might recover some of the velocity head during the expansion in it.

If this was in excess of the head lost during the expansion, the net results might be an increase in the discharge. Further, the suppression of the contraction by an entrance mouthpiece not only decreases the entrance loss but also reduces the turbulence of the subsequent flow in the pipe, thereby increasing the effect of a discharge mouthpiece. This will be discussed later under the heading of Combinations of Mouthpieces and Effect of Smooth Flow.

10. *Earlier Experiments with Entrance Mouthpieces.*—The results of other experimenters on entrance mouthpieces are not entirely consistent but give data of considerable importance. Table 3 gives in condensed form the results of the more important experiments. From this table it will be seen that adding a straight pipe to the mouthpiece used by Balch increased the discharge while in the experiments by Stewart the discharge was decreased. Furthermore, the coefficient of discharge increased with the velocity in the experiments by Balch and also in those by Davis and Balch but decreased in the experiments by Ellis, while Stewart found the coefficient to decrease at first and then to increase (not shown in Table 3). In the experiments recorded in this bulletin, the coefficient remained nearly constant. It will be observed that the mouthpieces used by the last four experimenters named in Table 3 have a throat diameter less than $1\frac{1}{4}$ in.; in fact in only one case is the throat diameter above 0.6 in. These mouthpieces give somewhat higher values for c than do mouthpieces of larger throat diameters. Perhaps the higher values for the coefficients of discharge for these small mouthpieces may be due to a lesser amount of turbulence; that is, it may be that the water flowing through a small mouthpiece is affected, or controlled more by the sides than in the case of a large mouthpiece, at least when the water enters or is received by the mouthpiece in a somewhat disturbed state of flow.

11. *Discharge Mouthpieces.*—As might be expected, the angle of the discharge mouthpieces influences the flow in a very different way from that of the entrance mouthpieces. Tables 5 and 6 and the curves of Figs. 7 and 8 show that when there is no mouthpiece on the entrance end of the short pipe, the coefficient of discharge for a discharge mouthpiece (attached to the pipe) diminishes rather rapidly as the angle of the mouthpiece increases, dropping somewhat

abruptly to practically no effect for an angle slightly greater than 20 degrees (40 degrees total angle).

It will be noted also by a comparison of Figs. 10 and 11 that increasing the length of the discharge mouthpiece beyond that corresponding to an area ratio of 1 to 2 has comparatively little effect on the rate of discharge. It may be, however, that an increase in length would show a greater effect for smaller angles than were used in these experiments, that is, for angles less than 5 degrees for an area ratio of 1 to 2 or less than 10 degrees for an area ratio of 1 to 3. But, it will appear (see article 12) that the size of the pipe, or the smallest area of the mouthpiece, is a much more important factor in the flow through a discharge mouthpiece than it is for an entrance mouthpiece. In other words, a discharge mouthpiece with a throat diameter of $\frac{1}{2}$ in. may give quite different results from that of a discharge mouthpiece having the same area ratio and the same angle of divergence but with a throat diameter of 6 in., at least when the water is received by the mouthpiece in a turbulent state of flow. The discharge mouthpiece with the small throat diameter (having a relatively large ratio of circumference to cross-sectional area) seems to be able to affect a greater percentage of the water flowing and thus regain more energy per pound of water discharged. Furthermore, the chances of having the water approach the discharge mouthpiece with smooth flow is greater in the case of the small pipe, hence perhaps the two causes work together, and for any given case they would be difficult to separate. For these reasons a comparison with earlier experiments on discharge mouthpieces for the purpose of extending the present experiments is apt to be misleading.

It is clear from Figs. 10 and 11 that the governing factor in the recovering of velocity head by means of a discharge mouthpiece attached to a short pipe, having a diameter of several inches or more, is the angle at which expansion begins—rate of expansion at the start—at least when the total angle of divergence is not less than 10 degrees and the area ratio not less than 1 to 2.

12. *Earlier Experiments with Discharge Mouthpieces.*—Table 4 gives, in condensed form, the results of the more important earlier experiments with discharge mouthpieces. These experiments seem to show the influence of the size of throat area as discussed above. The increase in the coefficient of discharge, c , with an increase in

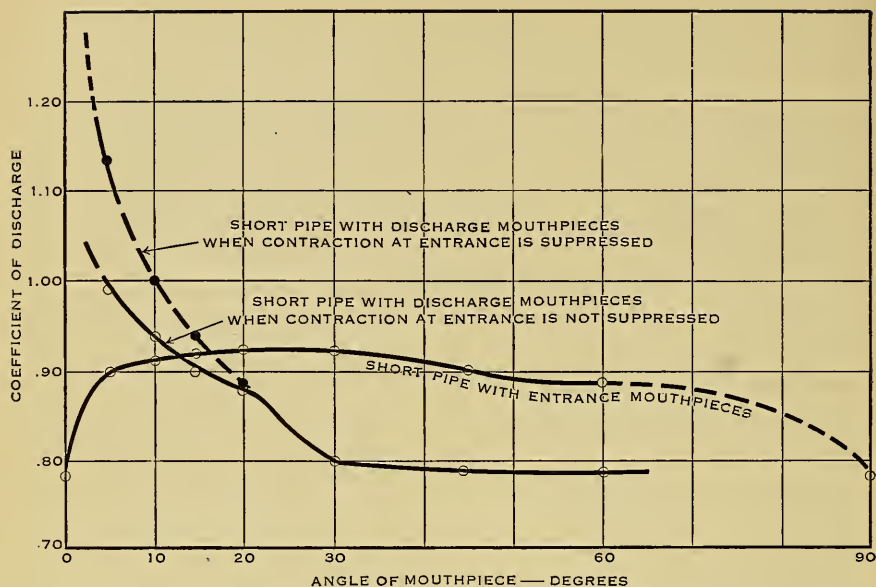


FIG. 10. RELATION BETWEEN COEFFICIENT OF DISCHARGE FOR THE SHORT PIPE WITH MOUTHPIECES AND ANGLE OF MOUTHPIECE FOR AREA RATIO OF 1 TO 2

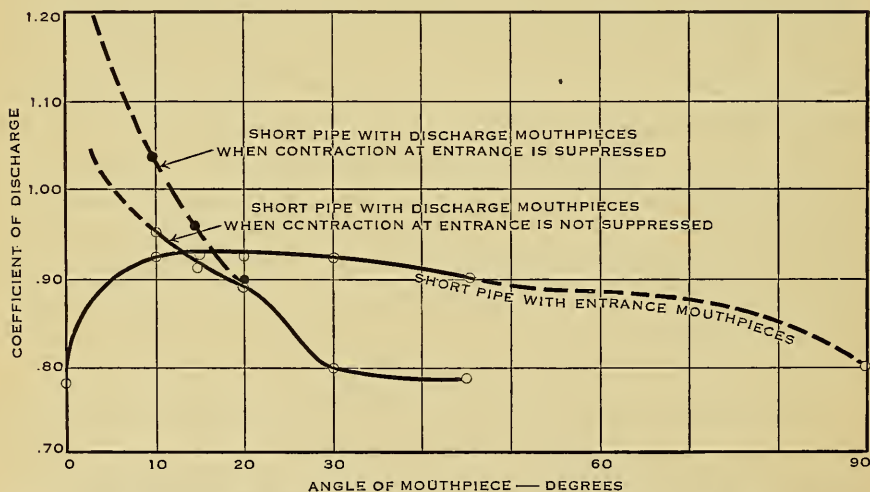


FIG. 11. RELATION BETWEEN COEFFICIENT OF DISCHARGE FOR THE SHORT PIPE WITH MOUTHPIECES AND ANGLE OF MOUTHPIECE FOR AREA RATIO OF 1 TO 3

the length of the discharge mouthpieces having small throat diameters is probably somewhat larger than would be obtained with mouthpieces having large throat diameters but with the same angles of divergence. As already noted in the experiments herein recorded the value of c increased very little for an increase in length corresponding to an increase in area ratio from 1 to 2 to 1 to 3 when the total angle of divergence was 20 degrees or more. It is clear from Table 4 that for discharge mouthpieces having small throat diameters and with the water in a state of smooth flow as it approaches the mouthpiece (entrance mouthpiece attached), an increase in length has a marked effect on the discharge. It will be noted that the greatest increase in c in the experiments reported by Francis occurred when the area ratio was increased from 1 to 2 to 1 to 5. It is probable that the increase in length would have been more noticeable in the present experiments for smaller angles of divergence, particularly for the smoother conditions of flow.

13. *Combinations of Mouthpieces and Effect of Smooth Flow.*—In order to get some measure of the influence of smooth flow upon the rate of discharge and the amount of velocity head recovered by a discharge mouthpiece, experiments were made using combinations of an entrance and a discharge mouthpiece when attached to the short pipe. The results are given in Table 7. It will be seen from this table that a discharge mouthpiece acts more effectively in recovering velocity head when a mouthpiece is attached to the entrance end. The following example will show this in the case of the 5-degree (1 to 2) discharge mouthpiece. This mouthpiece gave a coefficient of discharge of 0.99 (Table 5) when no mouthpiece was attached to the entrance end of the short pipe. From Table 7 it will be seen that the short pipe with a combination of the 5-degree (1 to 2) discharge mouthpiece and the 20-degree (1 to 2) entrance mouthpiece gave a coefficient of discharge of 1.34. This means that the 5-degree mouthpiece has a coefficient of discharge of 1.137 when used in combination with the 20-degree entrance mouthpiece. This value is obtained from the following steps: Attaching a 5-degree (1 to 2) discharge mouthpiece to the short pipe when no entrance mouthpiece is used is equivalent to raising the head on the inward projecting pipe from h to $1.59h$ as obtained by equating the rates of discharge,

$$0.99 a \sqrt{2gh} = 0.785 a \sqrt{2gH_D}, \text{ from which } H_D = 1.59h.$$

In like manner it is found that attaching a 20-degree (1 to 2) mouthpiece to the entrance end of the pipe when no mouthpiece is attached to the discharge end is equivalent to raising the head on the inward projecting pipe from h to $1.39h$ ($H_E = 1.39h$).

Now if both of these mouthpieces were attached to the short pipe, it might be expected that the head on the inward projecting pipe required to give the same discharge (equivalent head) would be $1.39h$ times 1.59 or $2.21h$ and that the coefficient of discharge for the combination, c_c , would be found from,

$$c_c a \sqrt{2gh} = 0.785 a \sqrt{2g(2.21h)} \text{ or, } c_c = 1.17^* \quad . \quad (7)$$

As already noted, however, the coefficient of discharge for this combination as found from experiment is 1.34 (Table 7) which corresponds to an equivalent head of $2.92h$. If all of this increase is attributed to the more efficient action of the discharge mouthpiece due to the fact that it receives the water in more nearly parallel streamlines (smooth flow), the result is an equivalent head for the 5-degree (1 to 2) discharge mouthpiece of $2.10h$ (Col. 7, Table 7) instead of $1.59h$, an increase of $0.51h$ due to smooth flow. Hence, the coefficient of discharge for the short pipe with the 5-degree (1 to 2) discharge mouthpiece, assuming smooth flow as the water approaches the mouthpiece, would be found from

$$c_d a \sqrt{2gh} = 0.785 a \sqrt{2g(2.10h)}, \text{ or } c_d = 1.137^\dagger \quad . \quad (8)$$

as compared with 0.99 . In other words, this particular mouthpiece gives an increase, due to smooth flow, of 14.9 per cent in the rate of discharge.

Tables 5 and 6 give the values of the coefficient of discharge for the short pipe with the various discharge mouthpieces attached when the water flowed through an entrance mouthpiece on its way to the discharge mouthpiece. These values are represented by the dash lines in Figs. 10 and 11. Even though the entrance mouthpiece used gives stream lines that are far from parallel, it is thought that a less turbulent condition of flow would seldom be found at least in a 6-in. pipe, and for velocities above 2 ft. per sec. Hence, the values of the

*See Table 7.

†See Col. 9, Table 5.

coefficient of discharge given by the dash lines are about the maximum to be expected. In each combination the entrance mouthpiece used was one giving about the minimum contraction.

Tables 7 and 8 also give the percentage gain in the coefficient of

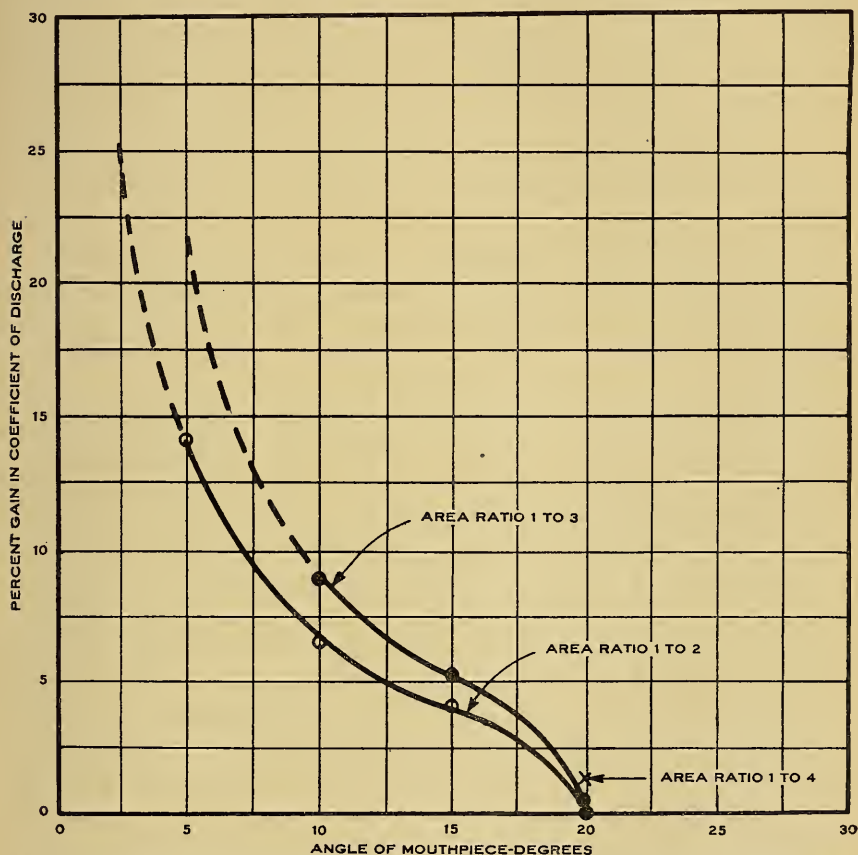


FIG. 12. RELATION BETWEEN PER CENT GAIN IN COEFFICIENT OF DISCHARGE FOR DISCHARGE MOUTHPIECES DUE TO ENTRANCE MOUTHPIECE, AND ANGLE OF DISCHARGE MOUTHPIECES

discharge for the various discharge mouthpieces due to the entrance mouthpiece. The relation of this gain to the angle of the discharge mouthpiece is shown in Fig. 12 for both series of mouthpieces. It will be noted that as the angle of the discharge mouthpiece increases

from 5 degrees (10 degrees total angle) the gain in the coefficient decreases rather rapidly, reaching zero for both series of mouthpieces at an angle of about 20 degrees (40 degrees total angle of divergence). Furthermore, this is the same angle at which a discharge mouthpiece rather abruptly ceases to regain any velocity head when no mouthpiece is used on the entrance end (see Figs. 10 and 11). It would seem, therefore, that a 20-degree discharge mouthpiece (total angle of divergence of 40 degrees), or one with a greater angle, allows such a turbulent condition of the water to develop that it is unable to recover any velocity head no matter how smooth the flow may be as the water approaches the mouthpiece. It is worth noting also that a comparison of the results in Table 5, Col. 10, and Table 6, Col. 10, indicates that the length of a discharge mouthpiece has a somewhat larger influence on the discharge when smooth flow exists than when the flow is more turbulent, and it is probable that this effect would have been more noticeable for smaller angles of divergence.

Attention has been called to the fact that the theoretical amount of velocity head which is possible of recovery by discharge mouthpieces having area ratios of 1 to 2, 1 to 3, and 1 to 4 is respectively 75, 88, and 94 per cent of the velocity head in the pipe. The effect of smooth flow may also be measured in terms of the increase in the amount of velocity head recovered by the discharge mouthpieces. For example, the coefficient of loss for the particular combination discussed above is 0.232 (Col. 5, Table 7) and the coefficient of loss for the pipe with the 20-degree (1 to 2) entrance mouthpiece only, is 0.165. Hence the loss of head in the discharge mouthpiece alone is 0.067 times the velocity head in the pipe. But 0.75 of the velocity head is the maximum amount possible of recovery, and since the mouthpiece lost 0.067 velocity heads, the amount recovered is 0.683 velocity heads which is 91 per cent of the maximum amount possible of recovery (Table 7, Col. 11). By a similar analysis it is shown that these same mouthpieces would have regained 58 per cent of the theoretical amount possible of recovery if the 5-degree (1 to 2) discharge mouthpiece had given the same action when in combination as it did when it was the only mouthpiece attached. That is, smooth flow allows a 5-degree (1 to 2) discharge mouthpiece to recover 33 per cent more velocity head than when the water approaches this mouthpiece in a turbulent state of flow (Table 5, Col. 11). The

relation between the increase in the velocity head recovered by the discharge mouthpieces due to an entrance mouthpiece, expressed in per cent of the maximum theoretical amount possible of recovery, and the angle of the discharge mouthpieces is shown in Fig. 11. It is probable that the two curves should have the same ordinate for a 15-degree mouthpiece. At least, the curve for the mouthpieces having an area ratio of 1 to 3 should not be above the other curve. If the same entrance mouthpiece had been used in each case, the difference would probably have been negligible. The curves, therefore, are drawn to give the same ordinate for the 15-degree mouthpiece.

In the foregoing discussion it has been assumed that adding a discharge mouthpiece to the short pipe will not change the state of flow in the pipe. This is probably true if the pipe has an entrance mouthpiece attached, suppressing the contraction. If, however, there is no mouthpiece on the entrance end, it appears that the influence of the discharge mouthpiece extends back into the pipe, producing, in effect, a mouthpiece whose smallest cross-sectional area is somewhat less than the area of the pipe and perhaps allowing the expansion of the stream in the pipe to be continuous with that in the mouthpiece. For example, the short pipe with a 10-degree (1 to 2, 20-degree total angle) discharge mouthpiece gave a coefficient of loss of 0.872 (Table 5), and since the short pipe without any mouthpiece attached gives a coefficient of loss of 0.62, the mouthpiece alone should give a loss of 0.252 times the velocity head in the pipe. But from Table 7 it will be seen that this same mouthpiece gives in combination with a 20-degree (1 to 2) entrance mouthpiece a coefficient of loss of 0.468, and after deducting 0.165, which is the loss for the short pipe with the 20-degree entrance mouthpiece, 0.303 is left as the coefficient of loss for the discharge mouthpiece alone. This is inconsistent with the value, 0.252, already established. A similar effect occurs with all the mouthpieces except the 5-degree. The explanation, as already suggested, seems to be that the coefficient of loss for the short pipe is less than 0.62 when a discharge mouthpiece is attached; that the influence of the mouthpiece is felt back into the pipe. This also helps to explain why the coefficients of discharge for mouthpieces with small throat diameters and connected by a short throat are relatively large.

Fig. 13 shows the relation between the coefficient of loss for the short pipe with entrance mouthpiece attached and the angle of mouth-

piece. It also shows the relation for the discharge mouthpieces alone as obtained by deducting the loss up to the mouthpiece. The values for the discharge mouthpieces alone are the larger ones as indicated in the example given. The values are given in Table 5. The number of velocity heads in the pipe recovered by any discharge mouthpiece would be the value obtained by subtracting the ordinate to the curve in Fig. 11 from 0.75 for an area ratio of 1 to 2 and from 0.88 for an area ratio of 1 to 3.

14. *Experimental Difficulties Encountered.*—During the early part of the investigation great difficulty was experienced in getting

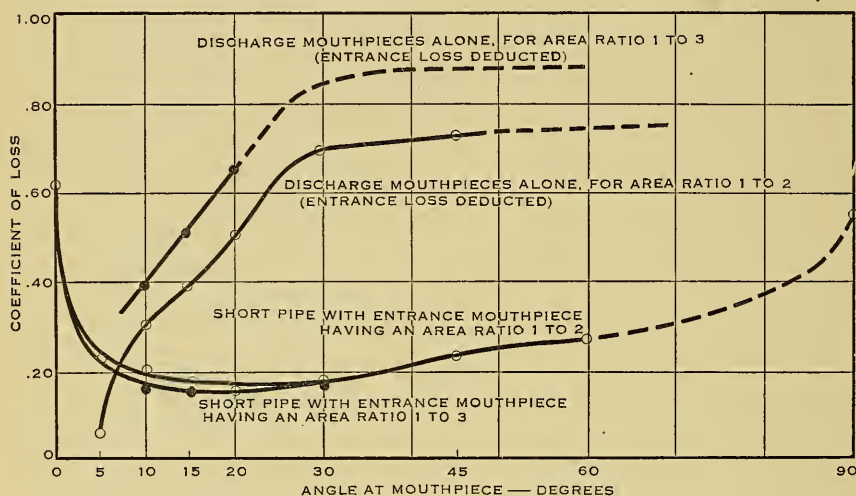


FIG. 13. RELATION BETWEEN COEFFICIENT OF LOSS AND ANGLE OF MOUTHPIECE

consistent results at the higher velocities, especially with mouthpieces having small angles of divergence. The head would fluctuate at times for apparently no good cause, and the value of the coefficient of discharge would vary considerably in successive experiments. It was noticed finally that vortices sometimes formed in the upstream compartment causing a "suck hole" as large as $1\frac{1}{2}$ in. in diameter at the water surface and tapering to a fine point some 10 or 12 in. below the surface. It was observed also that this vortex motion was more active and persistent when the unsteady conditions prevailed, but in no case did it appear to allow air to enter the pipe. In order to

prevent the formation of the vortex a float was made fitting closely in the upstream compartment. Grill work on the under part of the float extended deep enough to suppress the vortex before it could fully form. Results were more consistent after the float was used, and it was kept in use for the remainder of the experiments. It was at once noticed, however, that the coefficient of discharge for the longer discharge mouthpieces was lowered by the use of the float and considerable time was spent in attempting to measure the effect

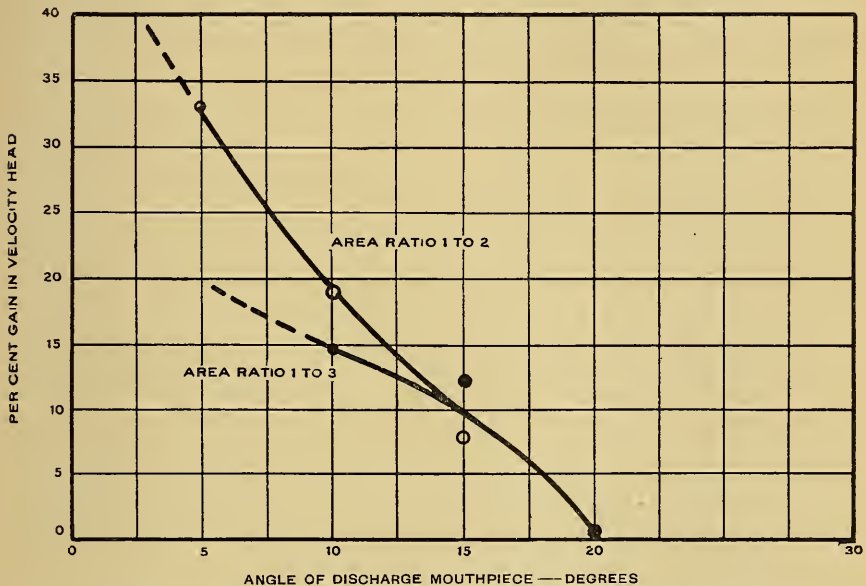


FIG. 14. RELATION BETWEEN GAIN IN VELOCITY HEAD RECOVERED BY DISCHARGE MOUTHPIECES DUE TO ENTRANCE MOUTHPIECE, EXPRESSED IN PER CENT OF THEORETICAL AMOUNT POSSIBLE OF RECOVERY, AND ANGLE OF DISCHARGE MOUTHPIECE

of the vortex. The experimental results clearly indicated that the vortex motion may increase the discharge by at least 2 per cent in the case of the discharge mouthpieces with the smaller angles. If air had entered the pipe, the discharge would, of course, have been decreased. It would appear, therefore, that the whirling motion of the water allowed it to enter the pipe with less contraction. With a 15-degree discharge mouthpiece, the vortex seemed to show no effect although

the vortex was not so large nor active as for the 5 and 10-degree mouthpiece. This suggests that perhaps the increase due to the vortex is caused more by the smoother flow of the water as it approaches the discharge mouthpiece, resulting in more efficient action by the mouthpiece (as already discussed), than by reduction in the entrance loss.

Another troublesome factor met with during the experiments was that of temperature changes of the water in the 2-in. pipes used as still basins for the hook gauges. The zero readings of the gauges were taken frequently during the experiments and were found to remain constant until the summer months when results became very erratic. After considerable time it was discovered that the trouble was due to the fact that the sun shone only on the pipe attached to the downstream compartment early in the afternoon, later shifting so as to shine only on the pipe attached to the upstream compartment. After correcting for the difference in temperature thus caused in the two still-basin pipes, consistent results were obtained. It was found, however, that the whole trouble could be avoided by frequently taking water from the tank and pouring it down the pipes.

15. *Conclusions.*—The preceding discussion has shown that the losses accompanying the flow of water depend largely upon the state of its motion which in turn is influenced by many factors, the effects of which in many cases can be but roughly estimated. While the results of these experiments tend to define the range of such effects for certain conditions of flow, additional experiments would be necessary to establish all the inferences which have been suggested. The following conclusions, however, seem justified:

a. As applying to conditions likely to be met in engineering practice, the value for the head lost at the entrance to an inward-projecting pipe (i. e. without entrance mouthpiece and not flush with wall of the reservoir) is 0.62 of the velocity head in the pipe ($0.62 \frac{v^2}{2g}$) instead of $0.93 \frac{v^2}{2g}$, as usually assumed. To put it in another form, the coefficient of discharge for a submerged short pipe with an inward-projecting entrance is 0.785 instead of 0.72 as given in nearly all books on hydraulics. Further, the lost head at the entrance to a pipe having a flush or square entrance is 0.56 of the velocity head in the pipe ($0.56 \frac{v^2}{2g}$)

instead of $0.49 \frac{v^2}{2g}$ as usually assumed. In other words, the coefficient of discharge for a submerged short pipe with a flush entrance is 0.80 instead of 0.82 as given by nearly all authorities.

b. The loss of head resulting from the flow of water through a submerged short pipe when a conical mouthpiece is attached to the entrance end, may be as low as 0.165 of the velocity head in the pipe ($0.165 \frac{v^2}{2g}$) if the mouthpiece has a total angle of convergence between 30 and 60 degrees and an area of ratio of end sections between 1 to 2 and 1 to 4 or somewhat greater. In other words, the coefficient of discharge for a submerged short pipe with an entrance mouthpiece as specified above is 0.915.

c. The loss of head which occurs when water flows through a submerged short pipe having an entrance mouthpiece varies but little with the angle of the mouthpiece if the total angle of convergence is between 20 and 90 degrees and if the area ratio is between 1 to 2 and 1 to 4 or somewhat more. The loss of head for any mouthpiece within this range would be approximately 0.20 of the velocity head in the pipe ($0.20 \frac{v^2}{2g}$). There is, therefore, little advantage to be gained by making an entrance mouthpiece longer than that corresponding to an area ratio of 1 to 2. Thus, an entrance mouthpiece with a total angle of convergence of 90 degrees and the length of which is only 0.2 of the diameter of the pipe gives approximately $0.20 \frac{v^2}{2g}$ for the loss of head.

d. The amount of velocity head recovered by a conical mouthpiece when attached to the discharge end of a submerged short pipe depends largely upon the angle of divergence of the mouthpiece, but comparatively little upon the length of the mouthpiece. This is true for lengths greater than that corresponding to an area ratio of 1 to 2 and for total angles of divergence of 10 degrees or more. The amount of velocity head recovered decreases rather rapidly as the angle of divergence increases from a total angle of 10 to 40 degrees. At or near 40 degrees the amount of velocity head recovered rather abruptly falls to approximately zero.

e. A conical discharge mouthpiece having a total angle of divergence of 10 degrees and an area ratio of 1 to 2, when attached to a submerged short pipe, will recover 0.435 of the velocity head in the pipe, which is 58 per cent of the theoretical amount possible of recovery.

f. The amount of velocity head recovered by a diverging or discharge mouthpiece when attached to a submerged short pipe is considerably more when a converging or entrance mouthpiece is also attached than it is when the entrance end of the short pipe is simply inward-projecting (no mouthpiece attached). This excess in the velocity head recovered diminishes rather rapidly as the angle of the discharge mouthpiece increases, and it becomes zero for a discharge mouthpiece having a total angle of divergence of approximately 40 degrees. This increase in the velocity head recovered is probably due to the effect of smooth flow in the pipe as the water approaches the discharge mouthpiece. The smooth flow allows the mouthpiece to recover more of the velocity head in the pipe than when a more turbulent flow exists; this increase amounts to as much as 33 per cent in the case of the discharge mouthpiece having a total angle of divergence of 10 degrees and an area ratio of 1 to 2.

While these conclusions are drawn from experiments on the flow of water through a particular short pipe having various entrance and discharge conditions, it is felt that the results of the experiments are applicable in a general way to a large variety of cases in engineering practice where the contraction and expansion of a stream of water occurs. A number of such cases are suggested in the introduction.

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*This reference was not found until after this bulletin had been written. The sizes of the mouthpieces and their angles of divergence are so small (about 0.6 in. throat diameter and 4 to 12 degrees total angle of divergence) and the velocity is so high (32 to 131 ft. per sec. throat velocity) that the results are not directly comparable with the results given in this bulletin. The article contains, however, much valuable information, emphasizing in particular the decrease in lost head when the water is caused to whirl or rotate about the axis of the tube as it enters the tube. The above article also refers to experiments of Banninger, published in "*Zeitschrift für das Gesamte Turbinwesen*," 1906, page 12. The results of these experiments have not been available for examination.

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UNIVERSITY OF ILLINOIS BULLETIN

ISSUED WEEKLY

Vol. XIV

MAY 28, 1917

No. 39

[Entered as second-class matter Dec. 11, 1912, at the Post Office at Urbana, Ill., under the Act of Aug. 24, 1912.]

EFFECTS OF STORAGE UPON THE PROPERTIES OF COAL

BY

S. W. PARR



BULLETIN No. 97

ENGINEERING EXPERIMENT STATION

PUBLISHED BY THE UNIVERSITY OF ILLINOIS, URBANA

PRICE: TWENTY CENTS

EUROPEAN AGENT

CHAPMAN & HALL, LTD., LONDON

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ENGINEERING EXPERIMENT STATION

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S. W. PARR

PROFESSOR OF APPLIED CHEMISTRY

ENGINEERING EXPERIMENT STATION

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EFFECTS OF STORAGE UPON THE PROPERTIES OF COAL

I. INTRODUCTION

The need of a thorough understanding of the conditions affecting the storage of bituminous coal is becoming more and more apparent. The demand for coal at certain seasons is so great that both mining and transportation facilities are taxed severely in meeting it. Provision has not been made for adequate and proper storage of bituminous coal either at the mines or at the distributing centers, and as a result of this lack there must be maintained throughout the year a sufficient number of operating mines to meet what may be termed the "peak load" which occurs during the winter months. At such times also there is often a shortage of cars, although a smaller number of cars even than is now available would be needed if the work of transportation could be more evenly distributed throughout the year. Mr. C. G. Hall, Secretary of the International Railway Fuel Association, has made an estimate showing that the number of excess mines over and above those which would be normally required to meet the demand, provided their work could be distributed evenly through the year, represents an investment in the United States of \$450,000,000. In addition, the extra coal cars, which must be at hand when the demand is heavy but which stand idle accumulating rust for the rest of the year, have cost the railways no less than \$105,000,000 more than would be necessary if the same tonnage could be hauled at an even rate. These wastes eventually affect the cost per ton of coal, which every one must pay as a contribution toward the capital investments.

The difficulties attending these conditions are accentuated by occasional abnormal demands such as are created upon the approach of any date for readjusting the wage scale with the accompanying possibility of a strike or lockout. For example, in addition to the normal excess demand during the winter months of 1915-16, a compilation of the published amounts of coal being stored by the various railway systems and larger users only, in view of a possible strike in April, aggregated over 3,000,000 tons.

The industrial disturbances do not include all of the serious considerations, however. If we consider the labor distresses that are

accentuated as a result of irregular employment, it at once appears that the problems involved are of great sociological as well as of economic interest.

The work here recorded is a continuation of certain studies, the results of which have been published in bulletin form by the Engineering Experiment Station under the titles of "The Weathering of Coal,"* "The Occluded Gases in Coal,"† and "The Spontaneous Combustion of Coal."‡

The subjects treated in these publications are of fundamental importance, and a thorough understanding of the principles involved under each subject is necessary before any adequate discussion can be undertaken of the problems connected with the storage of coal. The investigations described in Bulletin 38 of the University of Illinois Engineering Experiment Station, on "The Weathering of Coal" were conducted with car-lot samples of coal stored under various conditions. The data obtained covered a period of one year. The investigation was continued for an additional period of five years more, or for a total period of six years. The coal was then turned over to the power plant for steam generation, and boiler tests were made to establish the character of the various samples. These data, together with other facts bearing upon the general subject of coal storage, have accumulated to an extent which warrants their being brought together for record and discussion in this form.

Special acknowledgment is due Mr. J. M. Lindgren, Chemist, and Mr. F. H. Whittum, Assistant Chemist, for the analytical data accumulated beyond the first-year period. Their experience and skill in the matter of sampling and in the use of analytical and calorimetric methods have been especially noteworthy.

II. SUMMARY OF RESULTS

The facts established by this investigation may be briefly summarized as follows:

(1) Freshly mined coal is chemically very active. Certain constituents have a marked affinity for oxygen, with which they enter into combination at ordinary temperatures. While the extent of this reaction depends upon the variety of the coal and

*S. W. Parr and W. F. Wheeler, Univ. of Ill. Eng. Exp. Sta. Bul. 38, 1909.

†S. W. Parr and Perry Barker, Univ. of Ill. Eng. Exp. Sta. Bul. 32, 1909.

‡S. W. Parr and F. W. Kressman, Univ. of Ill. Eng. Exp. Sta. Bul. 46, 1910.

the amount of these active constituents, a very important factor is the fineness of division or the sum total of the superficial areas of the particles, and accessibility of oxygen to the mass.

(2) The actual loss of heat value resulting from storage is small. It is evident that upon mining out the coal from the bed certain volatile constituents of the marsh gas variety are set free. The heat values represented by such exudations are not great. The tendency to absorb oxygen from the air is also a marked characteristic of freshly mined coal. This is in reality a chemical process, and is accompanied by the generation of a small amount of heat, but these heat losses, compared with the total heat available in the coal, are insignificant. Indeed, it may be fairly questioned whether the heat losses are not more apparent than real since there is an increase of weight due to the absorption of oxygen. Such increase will in itself lower to a corresponding degree the indicated heat value per pound of coal.

(3) There is an increase of "fines" or slack resulting from storage, greater with some coals than with others. This, together with the saturation of the free burning constituent with oxygen, slows up the fire and gives the appearance of being lacking in heat value. However, with an increase of draft and a correct understanding of the combustion conditions to be maintained, a most excellent over-all efficiency can be secured even from coals which have been in storage for long periods.

(4) Bituminous coal can be stocked without appreciable loss of heat values provided the temperature is not allowed to rise above 180 degrees F. Any method of storage, to be successful, must either check or prevent the absorption of oxygen to such an extent that the generation of heat shall not proceed faster than the dissipation and loss of heat due to absorption or radiation.

(5) Underwater storage prevents loss of heat values, and is not accompanied by deterioration in physical properties, such as slacking. The water retained by the coal upon removal is substantially only that held by adhesion or capillarity.

(6) Dry storage is safer and more satisfactory if the fine material is screened out at the storage yard and lump only, preferably sized, is stocked.

It will be seen from this summary that the most serious part of the problem relates to the matter of spontaneous heating, and probably the least serious phase relates to deterioration and actual loss of heat values. It is certain that at the present time a better understanding of these difficulties has been reached, and there is reason for believing that this better understanding of the fundamental principles involved will lead to some practicable and safe procedure for the stocking of bituminous coal.

III. EFFECTS OF STORAGE UPON COAL

HEATING AND SPONTANEOUS COMBUSTION

It is a well established fact that freshly mined coal has a large absorptive capacity for oxygen. In Bulletin 32, "The Occluded Gases in Coal," it is made evident that this avidity for oxygen is most marked in the freshly mined coal, and after exposure to the air for four or five months an approach to the saturation point seems to be reached after which very little oxygen is taken on. A correct interpretation of this phenomenon is essential to an understanding of the spontaneous heating of coal piles. The natural conclusion would be to the effect that the oxygen has been simply absorbed or occluded; that it was a physical rather than a chemical change. The evidence, however, of all the more recent investigations goes to show that it is in fact a chemical combination and that it is accompanied by the generation of a small amount of heat. In Bulletin 46, "The Spontaneous Combustion of Coal,"* it is seen that at a temperature of from 35 to 40 degrees C (95 to 104 degrees F), and with free access of air, the amount of heat generated caused a rise in temperature of from 1 to 1½ degrees C per day. Porter and Ovitz† have measured the quantity of oxygen taken up by a sample of Franklin County coal and found it to be approximately 0.8 per cent of the weight of the coal. Moreover, there is only a very small amount of CO₂ formed. This is explained by the fact that the presence of certain unsaturated compounds allows the oxygen to enter the molecular structure of the coal, with which compounds the oxygen readily combines.

These references are a few of many that might be brought forward showing that at ordinary temperatures freshly mined coal unites

*Parr and Kressman, Univ. of Ill. Eng. Exp. Station Bul. 46, 1910.

†Journal Industrial and Engineering Chemistry, Vol. 2.

chemically with oxygen and that in the process there is generated a certain amount of heat.

1. *Oxidation of Pyrites*.—Much consideration has been given by various investigators to the role of iron pyrites in promoting the heating of coal. In the experiments carried out by Dennstedt and Bunz in 1908,* it appears that self-ignition may be brought about in the case of coals having only small amounts of pyritic sulphur. The conclusion is made, therefore, that the presence of iron pyrites is not an essential condition for spontaneous heating. Other investigators working along similar lines have reached the same conclusion. Still others seem to have evidence that pyritic sulphur is an active element in the case. A summary of opinions on this point is given as follows:†

“As to what part sulphur compounds, especially pyrite, play in the spontaneous ignition of coal, opinions differ greatly. Some believe pyrite to be the leading factor, while others believe it plays no part at all, or, if so, ascribe to it a position of minor importance and believe its action to be merely a subsidiary one. The oxidizing action of the air upon pyrite is, however, admitted, and the notion seems to be fairly general and well established that pyritic oxidation tends to raise the temperature of the coal. On the other hand, it is seen from the work of Fayol, Dennstedt and Bunz, Threlfall and others that coals containing pyrite in a quantity too insignificant to be noticed are very apt to ignite spontaneously. The Newcastle coal of New South Wales is also a very good example of this class of coals. Others, however, believe that the only influence of the pyrite is a mechanical one, in which the oxidation of the thin films of pyrite in the coal serves merely to break up the coal.”

Investigations of this type, having for their object a study of the processes of oxidation which occur at normal temperatures, are extremely important, especially in that phase of the work which seems to have fully established the fact of oxidation of the organic constituents of the coal. The conclusions are somewhat at fault, however, in assuming that as a consequence the pyritic oxidation is of little importance. It is true that a coal may heat seriously even though pyritic sulphur is absent. This does not constitute proof, however, that the presence of

*Zeit für Ang. Chemi., Vol. 21, pp. 1821-35, 1908.

†Parr and Kressmann, “The Spontaneous Combustion of Coal,” Univ. of Ill. Eng. Exp. Sta. Bul. 46, pp. 83, 1910.

pyritic sulphur in coals may not be equally, or even more largely, responsible for heating than the organic constituents.

This is clearly set forth in Bulletin 46.* On page 52, under "Iron Pyrites," a summarized statement is given as follows:

"The presence of sulphur in the form of iron pyrites is a positive source of heat due to the reaction between sulphur and oxygen. This may be conveniently referred to as the second stage in the process of oxidation. Here again rapidity of oxidation is directly dependent upon fineness of division. Since coals, as a rule, have a much higher earthy or ash content in the fine duff, and since iron pyrites is a large component of this material, it follows that the presence of dust or duff in all coals of the Illinois type is a positive source of danger."

However, in this summary the authors give first place in time and effect to the oxidation of the organic matter and consider that the activity of the pyrites waits somewhat upon the rise in temperature from such organic oxidation before action with sulphur reaches a serious phase. Special emphasis was laid upon the oxidation of sulphur as a source of heat, but the experiments did not specifically give direct evidence as to the temperature at which pyritic sulphur began to oxidize.

2. *Growth of Sulphates.*—Data on the oxidation of sulphur have recently been developed in connection with the study of variations in the determination of ash values† which have a bearing in this connection. The fact appears that the oxidation of sulphur is active at ordinary temperatures provided (a) that the pyritic iron be finely divided, and (b) that free moisture be present in sufficient amount to satisfy the reactions involved. For example, a certain series of bed samples of coal had been ground to 60-mesh and laboratory samples taken of about 75 grams which were placed in 4-ounce bottles with rubber stoppers. These samples were retained in the laboratory at room temperature from August, 1912, to April, 1913, at which time they were analyzed for sulphur in the sulphate or SO_3 form. For comparison, the original samples ground only to 10-mesh were similarly analyzed. The results are shown in Table 1.

*Parr and Kressmann, Univ. of Ill. Eng. Exp. Sta. Bul. 46, 1910.

†S. W. Parr, Ill. State Geo. Survey, Co-Op. Bul. 3, 1915.

TABLE 1
GROWTH OF SULPHATE
COMPARISON BETWEEN FINE AND COARSE LABORATORY SAMPLES

Lab. No.	County	H ₂ O	Total Sulphur	SO ₃ 60-Mesh Aug., '12	SO ₃ 60-Mesh April, '13	SO ₃ 10-Mesh April, '13
5365	Mercer.....	6.33	5.29	0.95	1.46	0.86
5367	Grundy.....	8.84	2.27	0.39	0.38	0.13
5370	Mercer.....	4.49	4.92	0.47	0.49	0.25
5372	Mercer.....	4.86	5.46	0.63	1.25	1.12
5376	Grundy.....	7.66	2.73	0.61	1.12	0.86
5388	La Salle.....	7.93	5.20	1.42	1.79	0.82
5381	La Salle.....	8.05	4.83	0.62	1.20	0.81

3. *Effect of Fineness of Division.*—From the results in Table 1 it is evident that oxidation of sulphur has occurred at ordinary temperatures. Five of the seven samples have had from 30 to 40 per cent of the total sulphur thus changed. The second and third samples in this table show no such oxidation. In the second the content of total sulphur is low and of this the actual sulphur in the pyritic form is, of course, still lower. Data on this point were not obtained. In the third sample the water content is lower than that in the other samples of the table. Whether this affords a valid explanation is uncertain. At any rate the point here emphasized is the fact that in the majority of the samples oxidation of the pyritic sulphur occurred in large amounts and at room temperatures. The next point was to determine what conditions were chiefly responsible in this reaction. The last column of the table affords some information. Here it is seen that with one exception the coarse or 10-mesh material had little or no indication of sulphur oxidation. To test further the effect of fineness of division, two of these original samples, about two pounds each, were sized and the sulphate sulphur determined for each size. The results are given in Table 2.

TABLE 2
SULPHATE SULPHUR AS FOUND IN VARIOUS SIZES OF COAL

Lab. No.	10-Mesh	20-Mesh	40-Mesh and Over
5372	0.91	0.89	1.43
5388	0.53	0.60	1.42

It is shown by this table that while substantially no increase in sulphate occurred up to the 20-mesh size, the increase was very

marked in the 40-mesh sample, though it should be said that the 40-mesh sample contained also all of the finer material passing through that sieve. This record further emphasizes the fact that oxidation of sulphur increases in activity as the size of particles is decreased and the superficial area in any given mass correspondingly increased.

4. *Effect of Moisture.*—In seeking an explanation for the lack of uniformity in behavior due to sizing alone, it was thought that possibly the amount of free moisture in the sample as well as the percentage of FeS_2 might play an important part. A number of samples were, therefore, selected in which the free moisture was low. In these cases the growth of sulphate in the laboratory sample was small as shown in Table 3.

TABLE 3
EFFECT OF LOW MOISTURE ON THE FORMATION OF SULPHATE
All Values on the Dry Coal Basis

Lab. No.	Moisture	Total Sulphur	Sulphate (SO_3) in Lab. Sample, 60-Mesh 75 Days in Sample Bottle	SO_3 in 3-Lb. Gross Sample. All Sizes up to $\frac{1}{4}$ Inch after 15 Days in Container	3-Lb. Gross Sample. All Sizes up to $\frac{1}{4}$ Inch after 75 Days	Sizing of Gross Sample after 75 Days		
						SO_3 in 10-Mesh Size	SO_3 in 20-Mesh Size	SO_3 in 40-Mesh Size
6399	1.74	3.65	.214	.200	.198	.176	.164	.215
6400	2.03	3.19	.218	.230	.228	.199	.195	.257

As affording further evidence on this general proposition, 28 samples of coal were selected from the various districts of the state, and sulphate determinations made on the laboratory samples ground to 60-mesh which had been in storage from the early part of 1912 until June, 1913. Unfortunately, the sulphate factors for the fresh coal are not available, but the table shows that in those samples in which both the water and the sulphur contents were high, there was a greater increase in the percentage of oxidized sulphur than in the case of samples in which water and sulphur contents were low. Note especially Nos. 5359-5389, inclusive, Table 4.

Doubtless, there are other circumstances connected with the oxidation of sulphur which are of interest, such as the presence of catalyzers, the source of oxygen for satisfying the conditions of the

TABLE 4

SULPHATE IN LABORATORY SAMPLES, STORAGE TIME FROM MARCH, 1912, TO JUNE, 1913, SHOWING CONDITIONS ATTENDING PRESENCE OF MOISTURE AND SULPHUR

Lab. No.	County	Coal Bed	H ₂ O	Total Sulphur Dry Coal	SO ₃ Dry Coal	Remarks
4699	Vermilion.....	6 N	2.30	2.44	0.18	Sulphate was determined June 2, 1913, on pulverized samples collected February to June, 1912.
4702	Vermilion.....	6 N	2.08	2.75	0.15	
4706	Vermilion.....	6 N	2.08	3.48	0.32	
4707	Vermilion.....	6 N	1.82	4.82	0.35	
4716	Vermilion.....	7 N	2.31	4.06	0.62	
4724	Vermilion.....	7 N	1.95	3.77	0.60	
4727	Vermilion.....	7 N	1.91	3.33	0.37	
4734	Vermilion.....	7 N	2.24	2.59	0.22	
4744	Vermilion.....	6 N	4.58	1.94	0.29	
4789	Franklin.....	6 S	3.88	0.52	0.02	
4811	Franklin.....	6 S	2.41	1.53	0.22	
4994	Saline.....	5 S	2.82	2.32	0.54	
5006	Williamson.....	6 S	3.25	1.11	0.12	
5011	Franklin.....	6 S	4.14	1.53	0.18	
5024	Saline.....	5 S	5.87	3.78	0.80	
5121	Williamson.....	6 S	3.73	1.42	0.11	
5122	Williamson.....	6 S	6.25	1.46	0.12	
5134	Williamson.....	6 S	6.25	1.24	0.10	
5224	Franklin.....	6 S	5.87	1.12	0.06	
5339	Mercer.....	1 N	3.21	5.66	0.98	
5359	Rock Island.....	1 N	4.84	6.56	2.14	
5361	Rock Island.....	1 N	5.74	4.56	1.08	
5362	Rock Island.....	1 N	4.96	5.26	1.15	
5364	Mercer.....	1 N	6.34	4.94	1.69	
5368	Grundy.....	2 N	7.08	3.04	0.59	
5369	Grundy.....	2 N	6.98	2.53	0.44	
5377	Grundy.....	2 N	7.83	4.00	1.34	
5389	La Salle.....	2 N	7.71	3.57	0.83	

reaction, the mercasite or pyrite form of the sulphide crystals, the method of distribution, whether in microscopic or massive aggregates, the character and influence of the associated material, and the segregation of sulphate crystals towards the finer materials. Indeed, the more thorough understanding of some of these points might contribute in a very practical way to our knowledge of the conditions which promote the heating of coal. Definite information along these lines depends upon further study. However, for purposes of this discussion it is of importance to note that at least two conditions, if existing, namely, fineness of division, and presence of moisture, will result in oxidation of the sulphur. Supplementary to this should be recalled the fact already developed in Bulletin 46 that the oxidation of 0.5 per cent of sulphur, or approximately less than $\frac{1}{2}$ of the amount present in the average Illinois coal, would produce sufficient heat to raise the temperature of the mass, not allowing for radiation losses, about 125 degrees F. If the initial temperature were 50 degrees F, an increase to 175 degrees would approach the danger

point. At about 180 degrees the activity reaches a stage owing to the greater rapidity of oxidation at that temperature, at which the chemical reaction quickly proceeds to the point where it becomes autogenous.*

In the discussion concerning the generation of heat from sulphur oxidation it is not intended to minimize the effect of oxidation of the organic matter as an initial source of heat, independent of the activity of the sulphur. In bituminous coals the two, doubtless, proceed independently, but where both activities exist together there is an acceleration of the reaction due to the rapid rise of temperature. In this manner there is a greater quantity of heat produced in a given period of time, and hence the coal mass comes more quickly and more positively to the autogenous or danger stage.

5. *Summary on Oxidation.*—Oxidation of the organic materials in freshly mined coal is active in all coals of the bituminous or lignitic type. The conditions which accelerate the action are increase of temperature and fineness of division.

Oxidation of the sulphur of iron pyrites is active provided the sulphide of iron is finely divided and there is sufficient moisture present to satisfy the reaction. The quantity of finely divided pyritic material will of course be greater in screenings than in lump, and since the ash content in screenings is from one to two times as great as in the lump, the quantity of pyritic sulphur is correspondingly greater.

The conditions to be observed in stocking coal so far as oxidation is concerned, are thus fairly well outlined. The enumeration of these conditions will be taken up later.

DETERIORATION

Experiments on the weathering of coal are described in Bulletin 38.† Car lots were stored in open and in covered bins and smaller lots of 100 to 200 pounds under water. The samples for determination of heat values were taken on the day of mining the coal, and thereafter at seven days, two months, six months and one year.

*Note the abrupt change in the direction of the curves at 80 degrees C, showing temperature rise, as in "Spontaneous Combustion of Coal," Univ. of Ill. Eng. Exp. Sta. Bul. 46, p. 28.

†Parr and Wheeler, "The Weathering of Coal," Univ. of Ill. Eng. Exp. Sta. Bul. 38, 1909.

All of the samples with the exception of the submerged samples, were continued in storage beyond the period covered by the results presented in Bulletin 38, for a total period of six years. Two additional sets of laboratory samples were taken; one after a total period of three years, and another at the end of six years. After the storage in the original bins for four years, both the covered and open lots were moved to a new location. After a period of six years, final laboratory samples were taken in connection with the cleaning up of the various lots and the making of boiler tests. Five standard boiler tests were made at the close of the period.

For greater convenience all of the analytical results for the entire period are presented in Tables 5 to 10, inclusive.

TABLE 5
VERMILION COUNTY NUT COAL

Lab. No.	Sample Taken after Mining	Dry Coal			B. t. u. Referred to Actual or Unit Coal	Decrease	
		Ash	Sulphur	B. t. u.		P. t. u.	Per Cent
STORED IN EXPOSED BINS							
1031	Same day.....	10.55	4.25	12991	14814	...	
1081	7 days.....	13.98	2.65	12412	14716	98	0.66
1240	2 months.....	14.21	2.47	12265	14577	237	1.60
1656	6 months.....	13.53	2.10	12396	14575	239	1.61
2088	1 year.....	13.62	2.82	12282	14498	316	2.13
4105	3 years.....	13.22	1.75	12018	14075	739	4.98
7298	6 years.....	13.06	1.58	11984	14000	814	5.49
STORED IN COVERED BINS							
1031	Same day.....	10.55	4.25	12991	14814	...	
1081	7 days.....	13.98	2.65	12412	14716	98	0.66
1249	2 months.....	13.08	2.13	12475	14604	210	1.42
1662	6 months.....	11.76	2.14	12571	14472	342	2.31
2094	1 year.....	13.52	2.72	12220	14403	411	2.77
4101	3 years.....	14.26	2.29	11691	13890	924	6.23
7372	6 years.....	9.84	1.57	1202	13867	947	6.39
STORED UNDER WATER							
1031	Same day.....	10.55	4.25	12891	14814	...	
1081	Same day as submerged	13.98	2.65	12412	14716	98	0.66
1647	6 months.....	15.37	3.34	12013	14524	290	1.96
2100	1 year.....	13.85	3.81	12231	14517	297	2.00

NOTE: For Unit Coal formula, see footnote to Table 12.

TABLE 6
WILLIAMSON COUNTY NUT COAL

Lab. No.	Sample Taken after Mining	Dry Coal			B. t. u. Referred to Actual or Unit Coal	Decrease	
		Ash	Sulphur	B. t. u.		B. t. u.	Per Cent
STORED IN EXPOSED BINS							
1090 } 1091 }	Same day.....	13.98	3.73	12499	14859	...	
1098	7 days.....	14.90	3.02	12341	14821	38	0.26
1246	2 months.....	14.32	4.12	12409	14835	24	0.16
1657	6 months.....	13.81	3.45	12455	14765	95	0.64
2090	1 year.....	11.88	2.73	12759	14734	125	0.84
4103	3 years.....	12.52	2.60	12503	14548	311	2.09
7374	6 years.....	12.86	2.20	12339	14406	453	3.04
STORED IN COVERED BINS							
1090 } 1091 }	Same day.....	13.98	3.73	12499	14859	...	
1098	7 days.....	14.90	3.02	12341	14821	38	0.26
1247	2 months.....	14.08	3.84	12378	14739	120	0.81
1663	6 months.....	13.06	3.60	12469	14644	215	1.45
2096	1 year.....	13.24	3.20	12428	14616	243	1.64
4100	3 years.....	13.93	2.93	12085	14323	536	3.60
7299	6 years.....	13.96	2.16	12103	14324	535	3.60
STORED UNDER WATER							
1090 } 1091 }	Same day.....	13.98	3.73	12499	14859	...	
1098	Same day as submerged	14.90	3.02	12341	14821	38	0.26
1648	6 months.....	15.65	3.12	12097	14673	186	1.25
2102	1 year.....	14.87	3.42	12251	14721	138	0.93

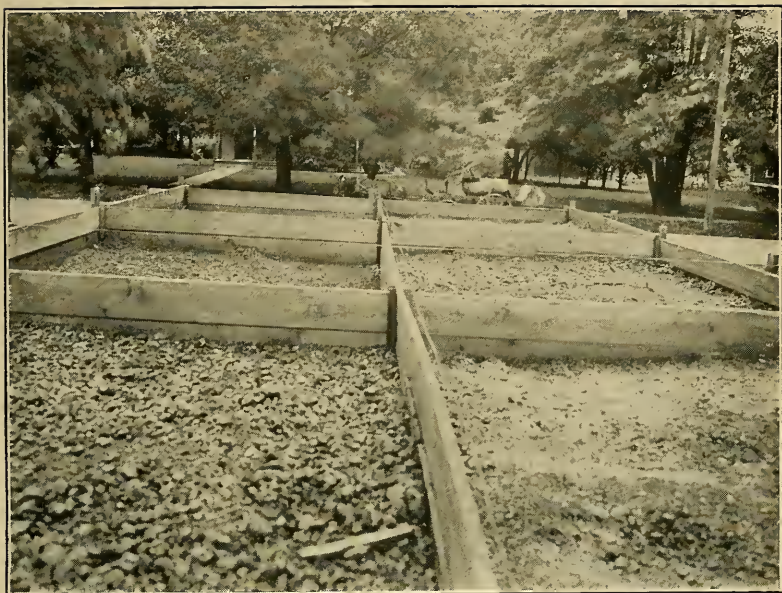


FIG. 1. OPEN STORAGE BINS

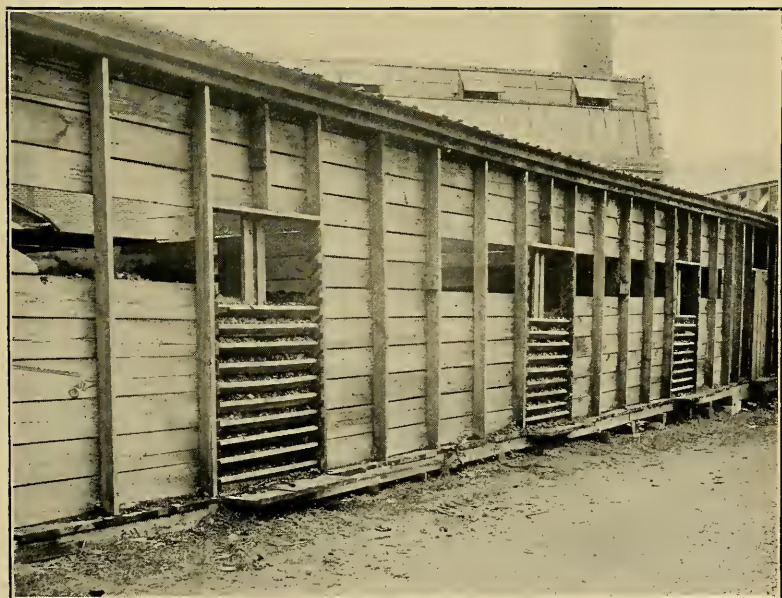


FIG. 2. COVERED STORAGE BINS

TABLE 7
SANGAMON COUNTY NUT COAL

Lab. No.	Sample Taken after Mining	Dry Coal			B. t. u. Referred to Actual or Unit Coal	Decrease	
		Ash	Sulphur	B. t. u.		B. t. u.	Per Cent
STORED IN EXPOSED BINS							
1078	Same day.....	17.87	5.75	11741	14773	...	...
1084	7 days.....	16.63	5.10	11800	14571	202	1.37
1248	2 months.....	17.45	4.66	11626	14497	276	1.87
1658	6 months.....	16.03	4.91	11798	14444	329	2.23
2086	1 year.....	14.97	4.68	11860	14307	466	3.15
4107	3 years.....	15.55	4.13	11621	14102	671	4.54
7277	6 years.....	14.10	3.76	11180	13813	960	6.49
STORED IN COVERED BINS							
1078	Same day.....	17.87	5.75	11741	14773	...	...
1084	7 days.....	16.63	5.10	11800	14571	202	1.37
1250A	2 months.....	16.08	5.03	11912	14600	173	1.17
1250B	2 months.....	17.57	5.01	11626	14535	238	1.61
1664	6 months.....	16.30	4.52	11682	14336	437	2.96
2092	1 year.....	15.99	4.65	11589	14165	608	4.12
4098	3 years.....	17.34	4.78	10681	13278	1495	10.11
7370	6 years.....	11.83	1.45	11971	13767	1006	6.81
STORED UNDER WATER							
1078	Same day.....	17.87	5.75	11741	14773	...	...
1084	Same day as submerged	16.63	5.10	11800	14571	202	1.37
1649	6 months.....	15.90	4.21	11854	14461	322	2.18
2098	1 year.....	15.95	5.11	11851	14503	270	1.83

TABLE 8
VERMILION COUNTY SCREENINGS

Lab. No.	Sample Taken after Mining	Dry Coal			B. t. u. Referred to Actual or Unit Coal	Decrease	
		Ash	Sulphur	B. t. u.		B. t. u.	Per Cent
STORED IN EXPOSED BINS							
1032	Same day.....	17.88	2.35	11937	14888	...	...
1080	7 days.....	13.98	2.87	12414	14726	162	1.09
1082	7 days.....	13.69	2.29	12507	14759	129	0.87
1238	2 months.....	15.73	2.53	11958	14497	391	2.63
1239	2 months.....	14.69	2.90	12178	14578	310	2.08
1653	6 months.....	15.63	2.44	11969	14487	401	2.69
2089	1 year.....	14.46	2.24	12006	14304	584	3.92
4108	3 years.....	15.95	1.98	11229	13625	1263	8.48
7369	6 years.....	13.79	3.85	11210	13275	1613	10.83
STORED IN COVERED BINS							
1032	Same day.....	17.88	2.35	11937	14888	...	
1080	7 days.....	13.98	2.87	12414	14726	162	1.09
1082	7 days.....	13.69	2.29	12507	14759	129	0.87
1241	2 months.....	15.26	2.51	12124	14608	280	1.88
1659	6 months.....	14.51	2.25	12071	14391	497	3.34
2095	1 year.....	15.36	2.42	11797	14225	663	4.46
4102	3 years.....	14.43	2.26	11199	13329	1559	10.47
STORED UNDER WATER							
1032	Same day.....	17.88	2.35	11937	14888	...	
1080	Same day as submerged	13.98	2.87	12414	14726	162	1.09
1082	Same day as submerged	13.69	2.29	12507	14759	129	1.87
1644	6 months.....	13.87	2.32	12270	14514	374	2.51
2101	1 year.....	13.55	2.71	12283	14483	405	2.72

TABLE 9
WILLIAMSON COUNTY SCREENINGS

Lab. No.	Sample Taken after Mining	Dry Coal			B. t. u. Referred to Actual or Unit Coal	Decrease	
		Ash	Sulphur	B. t. u.		B. t. u.	Per Cent
STORED IN EXPOSED BINS							
1089	Same day.....	14.13	3.17	12426	14782	...	
1099	7 days.....	14.37	3.34	12287	14666	116	0.78
1244	2 months.....	15.66	2.67	12133	14701	81	0.55
1654	6 months.....	13.76	2.84	12342	14597	185	1.25
2091	1 year.....	13.77	2.75	12328	14579	203	1.37
4106	3 years.....	13.25	2.26	13335	14470	312	2.11
7371	6 years.....	13.42	2.23	12077	14196	586	3.96
STORED IN COVERED BINS							
1089	Same day.....	14.13	3.17	12426	14782	...	
1099	7 days.....	14.37	3.34	12287	14666	116	0.78
1245	2 months.....	12.62	2.98	12608	14705	77	0.52
1660	6 months.....	13.60	3.03	12372	14610	172	1.16
2097	1 year.....	13.43	2.72	12385	14582	200	1.35
4099	3 years.....	13.07	2.66	12146	14230	552	3.73
7281	6 years.....	12.64	2.19	12153	14144	638	4.31
STORED UNDER WATER							
1089	Same day.....	14.13	3.17	12426	14782	...	
1099	Same day as submerged	14.37	3.34	12287	14666	116	0.78
1645	6 months.....	14.38	3.54	12262	14645	137	0.93
2103	1 year.....	13.60	2.97	12447	14698	84	0.57

TABLE 10
SANGAMON COUNTY SCREENINGS

Lab. No.	Sample Taken after Mining	Dry Coal			B. t. u. Referred to Actual or Unit Coal	Decrease	
		Ash	Sulphur	B. t. u.		B. t. u.	Per Cent
STORED IN EXPOSED BINS							
1079	Same day.....	17.13	4.92	11752	14604	...	
1085	7 days.....	17.04	4.47	11684	14481	123	0.84
1242	2 months.....	17.22	5.00	11645	14488	116	0.79
1655	6 months.....	17.02	4.54	11526	14281	323	2.21
2087	1 year.....	17.25	4.54	11153	13853	751	5.14
4104	3 years.....	15.94	4.03	11131	13565	1039	7.11
7373	6 years.....	14.95	3.32	11326	13604	1000	6.85
STORED IN COVERED BINS							
1079	Same day.....	17.13	4.92	11752	14604	...	
1085	7 days.....	17.04	4.47	11684	14481	123	0.84
1243	2 months.....	18.33	4.70	11414	14404	200	1.37
1661	6 months.....	17.30	4.67	11466	14263	341	2.33
2093	1 year.....	17.06	4.73	11248	13944	660	4.52
4097	3 years.....	18.71	4.86	10325	13072	1532	10.48
7279	6 years.....	17.39	3.81	10497	13025	1579	10.81
STORED UNDER WATER							
1079	Same day.....	17.13	4.92	11752	14604	...	
1085	Same day as submerged	17.04	4.47	11684	14481	123	0.84
1646	6 months.....	19.86	5.60	11127	14372	232	1.59
2099	1 year.....	18.27	4.81	11479	14478	126	0.86

6. *Indicated Heat Losses.*—An inspection of the tables shows that the heating value decreases most rapidly during the first week after mining and continues to decrease more and more slowly for an indefinite time. One per cent is about the average loss for the first week, and an additional loss of two or three per cent may occur by the end of the first year. At the end of the six-year period the indicated losses in some cases equal nearly eleven per cent.

These losses may be ascribed to three distinct causes:

- The escape of combustible gases.
- The absorption of oxygen.
- The increase in weight of the organic or combustible portion of the coal.

7. *Escape of Combustible Gases.*—From certain experiments given in a former bulletin* there is positive evidence that combustible gases exude from freshly mined coal. The extent of this loss, however, is very small. Porter and Ovitz† have carried out a quantitative measurement of such escaping gases and estimate in the case of a coal from Benton, Franklin County, Illinois, the total loss of heat values in a period of seventeen months to be 0.16 per cent. As this was an extreme case for this variety of coal it is evident that the heat loss due to the exudation of combustible gases is practically negligible.

8. *Absorption of Oxygen.*—In Bulletin 32‡ on the occluded gases in coal it is shown that freshly mined coal has a marked avidity for oxygen. It has been shown¶ that the union of oxygen with the coal is a chemical combination and not a simple absorption. While we should expect such chemical action to be accompanied by the generation of heat and consequently by a reduction in the heat value of the coal, there is little information available as to the extent of such loss.

Porter and Ralston§ show the positive formation of heat from oxygen combination but do not present any data as to the amount of this loss. Lamplough and Hill** have attempted to measure the amount of heat, and their results show an average for English coals of 3.3 calories for each cubic centimeter of oxygen absorbed. Winmill and Graham†† have carried the same line of experimentation further and have modified slightly the factor for the heat generated, their result showing 2.1 calories per cubic centimeter of oxygen absorbed. The difference is, for the purpose of the present discussion, not material since the desired result is the approximate heat loss due to the oxygen combinations at ordinary temperatures. The maximum absorption under the most favorable circumstances ranges from 7 to 8 cc. oxygen per gram of coal as recorded in the experiments by Lamplough and

*Parr and Wheeler, "The Deterioration of Coal Samples," Univ. of Ill. Eng. Exp. Sta. Bul. 17, p. 33.

†U. S. Bureau of Mines, Technical Paper No. 2, 1910.

‡Parr and Barker, "The Occluded Gases in Coal," Univ. of Ill. Eng. Exp. Sta. Bul. 32.

¶Parr and Hadley, "The Analysis of Coal with Phenol as a Solvent," Univ. of Ill. Eng. Exp. Sta. Bul. 76.

§U. S. Bureau of Mines Tech. Paper 65, p. 8, 1914.

**Trans. Inst. Mining Engrs., Vol. 45, p. 629, 1913.

††Winmill and Graham, "The Absorption of Oxygen by Coal," Colliery Guardian, Sept. 11-18, 1915.

Hill and also by Winmill and Graham. Porter and Ralston found in the case of a Franklin county, Illinois, coal exposed for five months at ordinary temperatures to pure oxygen, an absorption at the rate of 5.3 cc. per gram of coal. This is approximately 10 calories per kilo of coal or about 0.12 per cent of the heat value of the coal. The loss of heat thus represented is so small that it is negligible as a factor effecting deterioration in the quality of the coal. It may, however, possess significance in the matter of spontaneous heating. This phase of the subject will be hereinafter discussed.

9. *Increase in Weight.*—It is evident that, if in the processes which attend the weathering of coal there is an increase in weight of the coal substance, the indicated heat losses are more apparent than real. For example, if at the beginning of the storage period a pound of the unit coal substance shows a value of 14,700 B.t.u., and at the end of the period the original pound has increased in weight by absorption or additions, say 5 per cent, then the heat value per pound of the resulting material will be 14,000 B.t.u., thus indicating an apparent loss of 4.76 per cent. Evidence from many sources has accumulated to show that coal exposed to air or oxygen increases in weight. In the experiments described by Parr and Hadley* it is shown that under certain conditions in the residue insoluble in phenol, which is regarded as the degradation product of the cellulose constituent, there is as much as 3 per cent increase in weight due to the taking up of oxygen. This work further shows that such additional oxygen is chemically combined and not merely absorbed. Other experimenters have presented evidence to the effect that coal increases in weight. Somermeier† has shown an increase due to oxidation for an Illinois coal 2.47 per cent. Porter and Ralston‡ show also a measurable increase in weight due to oxygen absorption. In their experiments on oxidation at various temperatures they note that Illinois and Pittsburgh coals “show increases of weight up to 260 degrees in spite of the loss of carbon and hydrogen in CO_2 , CO and H_2O .” Study of some of the values presented in Table 11 seems to give

*“The Analysis of Coal with Phenol as a Solvent,” Univ. of Ill. Eng. Exp. Sta. Bul. 76.

†Professor Somermeier was doubtless the first to suggest that the indicated decrease in heat values was in reality the result of an increase in weight. N. W. Lord and E. E. Somermeier, U. S. Geol. Surv. Bul. 323, p. 22.

‡U. S. Bureau of Mines, Tech. Paper 65, p. 20-22, 1914.

further evidence of an increase in weight of the organic or combustible part of the coal. Some of these data with a discussion of their bearing on this point are presented under the following topic.

10. *Decrease of Ash Percentages.*—A study of the relative ash values corresponding to the various stages of indicated heat losses reveals a consistent lowering of the percentages of ash. This result obviously is normal since the actual increase in the organic constituents of coal during storage with the corresponding increase in the weight of any given sample must result in a relatively lower amount of ash, or an apparent decrease in ash.

Of course, the exact duplication of these theoretical conditions was impossible in these experiments. The oxidation of the sulphur varied and the leaching out of the soluble sulphates would alter the ash values in a corresponding degree. Other variables might enter, such as irregularities in sampling or the possible accumulation of foreign or earthy matter. Notwithstanding all these possibilities of inaccuracy there is a striking consistency of results with reference to the decrease of indicated ash percentages as the process of weathering proceeded. The ash values of the several masses taken at the beginning and at the close of the six-year period are presented in Table 11.

TABLE 11
INDICATED ASH AT THE BEGINNING AND AT THE END OF SIX YEARS IN STORAGE
IN OPEN BINS

No.	Coal and County	Average Ash Values as Determined by Analysis of 3 Samples (1) at the Mine (2) at Unloading of Car (3) after 2 Months	Average Ash Values after 6 Years as Shown by Analysis of 2 Samples (1) from Open Bins (2) from Covered Bins	Indicated Decrease in Ash
1	Nut—Sangamon.....	17.01	12.96	4.05
2	Nut—Vermilion.....	12.95	11.45	1.50
3	Nut—Williamson.....	14.32	13.41	0.91
4	Screenings—Sangamon..	17.43	16.17	1.26
5	Screenings—Vermilion ..	15.20	13.79	1.41
6	Screenings—Williamson .	14.19	13.03	1.16

Inspection of the ash values presented in Tables 5 to 10, inclusive, further emphasizes the results shown by Table 11. The factors for the submerged coal have not been included in Table 11. Although

these samples were under test for only a year, they show a constancy in the ash values and an absence of change, which is in keeping with the uniformity of heat values credited to the unit coal substance.

The increase in weight of the coal in storage by addition to the organic constituents is further illustrated by the values presented in Table 12. The values presented in Column 1 represent the heat

TABLE 12

INCREASES IN WEIGHT OF COAL DURING STORAGE, IN RELATION TO THE INDICATED DECREASE IN HEAT VALUE

No.	Coal and County	B. t. u. per Lb. of Fresh Coal Average of 3 Samples Taken (1) at Mine (2) at Un- loading of Car (3) after 2 Mos. Storage	B. t. u. per Lb. after 6 Years Open Storage, Dry Coal Basis	Ash as Weighed Dry Coal Basis, with Heat Values as in Col. 2	Sulphur in Dry Coal after 6 Years Open Storage	Ash Plus Addi- tive Mate- rial Ac- quired in 6 Years Open Storage	Ash as in Col. 3 but Cor- rected for Sulphur as in For- mula for Unit Coal	Show- ing Amount of Addi- tive Mate- rial by Dif- ference
		1	2	3	4	5	6	7
1	Nut— Sangamon.....	14,614	11,180	14.10	3.76	23.5	17.3	6.2
2	Nut— Vermilion.....	14,700	11,984	13.06	1.58	18.5	14.9	3.5
3	Nut— Williamson.....	14,838	12,339	12.86	2.20	16.9	15.1	1.8
4	Screenings— Sangamon.....	14,524	11,326	14.95	3.32	21.9	17.9	4.0
5	Screenings— Vermilion.....	14,717	11,210	13.79	3.85	23.8	17.0	5.2
6	Screenings— Williamson.....	14,716	12,077	13.42	2.23	17.2	15.7	1.5

For discussion of Unit Coal and Corrected Ash see Univ. of Ill. Eng. Exp. Sta., Bul. 37, p. 33.

$$\text{Unit B. t. u.} = \frac{\text{Indicated (dry) B. t. u.} - 5000 \text{ sulphur}}{1.00 - (1.08) + 22/40 \text{ sulphur}}$$

$$\text{Corrected (dry) ash} = \text{ash as weighed} \times 1.08 + 22/40 \text{ sulphur.}$$

value for the unit coal when fresh, as found by averaging the unit coal values for three samples taken at the mine, at the time of unloading, and after two months in storage. The calorific values of the dry coal in open storage at the end of six years are shown in Column 2. The accompanying ash and sulphur values are shown in Columns 3 and 4. Column 5 shows the percentages of ash or inert material which must be present in order that the fresh unit coal

values of Column 1 may after six years drop to the values shown in 3. These percentages are derived by the formula,

$$100 - \frac{\text{B.t.u. per pound after storage}}{\text{B.t.u. per pound of fresh coal}}$$

The values in Column 6 are the corrected ash values for the coal after six years in open storage.*

The difference between the apparent corrected ash values of Column 6 and the required ash for producing the values shown in Column 2 is a measure of the additive material which is assumed to have been taken up by the organic constituents as absorbed oxygen or hydroxyl additions. These additions bear a certain general relation to the indicated decrease of ash shown in Table 11 and, of course, thus bear a more direct relation to the indicated deterioration percentages (see Tables 5 to 10).

These facts taken together, therefore, seem to afford added basis for the statement that the actual losses of heat values in stored coal are apparent rather than real, and that the true heat losses are those due to escaping combustible gases and to the heat generated by direct combination of oxygen both of which have been shown to be practically negligible.

IV. MOISTURE VALUES FOR WEATHERED COAL

The moisture values for the stored coal at the end of the six-year period are presented in Table 13. Some recent work on the Properties of the Water in Coal† by Porter and Ralston suggests a relation between the type of coal and the amount of "inherent" or hygroscopic moisture retained by the coal upon air-drying. It might be argued, therefore, that if the coals here considered during storage altered in type, possibly by a reversion toward the lignitic form, then a correspondingly high percentage of the moisture should be retained on air drying. The values shown in the table are without significance as far as this theory is concerned. However, the data should be

*Corrected Ash (dry) = ash $\times$ 1.08 + $\frac{22}{40}$ $\times$ sulphur (see formula for Unit Coal, footnote to Table 12).

†H. C. Porter and O. C. Ralston, U. S. Bureau of Mines, Technical Paper 113, 1916.

presented for other reasons. For example, the high percentage of total moisture is in a general way characteristic of the several kinds of coal in that it is affected by the degree of subdivision or disintegration which has taken place. As may be seen by reference to the discussion concerning the slacking of these coals, the coals which have undergone the greatest disintegration have the highest percentage of total moisture present. This result, however, is to be expected and is due to physical rather than to chemical action.

V. SUMMARY OF CHEMICAL STUDIES

The results presented in the foregoing tables and the data analyzed in the discussions may be summarized as follows:

(1) After a period of one year, the indicated loss of heat values is relatively low, averaging about 3 or $3\frac{1}{2}$ per cent.

(2) Different coals vary in indicated heat losses, those from the southern Illinois districts showing less change than those from the central part of the state. Exposure beyond a period of one year accentuates this difference between coals from different localities. The denser coals, such as those from Williamson county, undergo but little additional change, while the coals from the northern parts of the state show a decrease in the indicated heat values. The difference, however, does not exceed about 10 per cent.

(3) Deterioration is consistently greater in the case of screenings than in that of screened nut.

(4) Since the heat values are referred to the unit coal basis, that is, the moisture, corrected ash, and sulphur free material, and since an actual loss of heat values by ordinary processes of oxidation would result in the formation of CO_2 and H_2O , both volatile under the conditions, it follows that the heat losses are largely relative, since they must result from a relative increase in weight of the organic substance of the coal. For example, any increase in weight, as by the addition of oxygen to the chemical structure of the organic material, would result in a lower indicated heat value per pound of the unit substance as compared with the heat value per pound before such addition had occurred. Similarly, changes in the sulphur combination due to oxidation would increase the weight of the

coal in a manner not taken into consideration in deriving the "corrected" ash values. Thus, where FeS_2 becomes $2\text{FeSO}_4 + 7\text{H}_2\text{O}$, the relative weights for sulphur are in the ratio of 1:7; that is, the content of sulphur compounds as the result of the new combinations has increased in weight seven times.

(5) Storage in *open bins* shows quite consistently a *lower* percentage of loss of heat value per pound than *is the case with storage under cover*. This is easily understood when it is considered that the additional material formed, due to the oxidation of the sulphur, is soluble and tends to leach out under the long continued application of water resulting from exposure to the elements. For this reason, therefore, the increase in weight is greater in the case of the coal under cover than in that of the coal exposed in open bins. The resulting unit coal values should, therefore, be higher for the coal stored in open bins, or the leached coal, than for the coal stored under cover or the unleached coal. Comparisons as to the relative values in covered and uncovered bins should, however, be made only up to and including the three-year period. The removal to new locations of the bins and piles shortly before the three-year old samples were taken resulted in the reconstruction of the covered bins with flat and leaky roofs through which the water had more or less free access to the coal.

(6) The extent of oxidation or increase in weight is a function of the character of the coal. The coal in which the cellulose residuum seems to predominate has the greater avidity for oxygen, and the coal in which the resinic residuum predominates is less affected. The tabulation of the samples, therefore, is in the order of such activity, namely, Sangamon, Vermilion, and Williamson counties. This feature is consistent with the studies in the absorptive capacities for oxygen of various coals as carried on by Porter and Ralston and is especially of interest in connection with the studies of Dr. Hadley on the relative avidity for oxygen of the cellulose residuum as compared with the absorptive capacity of the resinic bodies, separation into these two type components being affected by means of phenol.*

*"The Analysis of Coal with Phenol as a Solvent," Univ. of Ill. Eng. Exp. Sta. Bul. 76, p. 21, 1914.

According to the results derived by Dr. Hadley, the cellulose residue had a far greater avidity for combination with oxygen than the resinic material.

TABLE 13

PERCENTAGE OF MOISTURE IN WEATHERED COAL, LOSS DUE TO AIR-DRYING, AND THE AMOUNT RETAINED BY THE AIR-DRIED SAMPLES

Table No.	Lab. No.	Coal, County and Condition	Total Moisture	Loss Due to Air-Drying	Moisture Retained in the Air-Dried Sample
1	7369	Screenings Vermilion Exposed Bin	17.70	14.15	4.14
2	7370	Nut Sangamon Covered Bin	19.95	16.54	4.09
3	7371	Screenings Williamson Exposed Bin	12.01	8.59	3.74
4	7372	Nut Vermilion Covered Bin	17.79	14.50	3.85
5	7373	Screenings Sangamon Exposed Bin	19.26	15.79	4.12
6	7374	Nut Williamson Exposed Bin	9.77	7.05	2.93
7	7277	Nut Sangamon Exposed Bin	20.35	17.01	4.03
8	7279	Screenings Sangamon Covered Bin	22.04	19.36	3.33
9	7298	Nut Vermilion Covered Bin	20.02	16.01	4.77
10	7299	Nut Williamson Covered Bin	12.54	9.45	3.30

VI. CHANGES IN PHYSICAL PROPERTIES

11. *Sizing Test*.—The extent of the disintegration or “slacking” which takes place in connection with the storing of coal is a matter of considerable importance because of the effect upon combustion on the grates where a large amount of finely divided material is present. In order to determine the effect of storage upon slacking the sizing tests were continued to cover the entire period of six years. The

tests were made with a revolving screen with round perforations, as shown in Fig. 3.

Three sizing tests were made: one at the time the coal was placed in storage, one after a period of eighteen months, and one at the end of six years. It should be recalled that approximately three years before the last screening tests the storage piles were removed by wagon to a new location involving a double handling. Since the

TABLE 14
RESULTS OF SIZING TESTS ON NUT COAL STORED IN OPEN BINS

Round-Hole Screen		Original Sizes		In Storage for 1½ Years		In Storage for 6 Years	
Through Inches	Over Inches	Per Cent	Cumulative Per Cent	Per Cent	Cumulative Per Cent	Per Cent	Cumulative Per Cent
SANGAMON COUNTY							
3	1	89.4		64.3		30.9	
1	¾	4.1	93.5	6.9	71.2	9.6	40.5
¾	½	3.5	97.0	8.4	79.6	15.9	56.4
½	¼	1.2	98.2	3.2	82.8	8.5	64.9
¼	⅛	0.6	98.8	4.0	86.8	3.2	68.1
⅛	0	0.6	99.4	7.4	94.2	17.0	85.1
0		0.6	100.0	5.8	100.0	14.9	100.0
Total		100.0		100.0		100.0	
Average Diameter ...			1.854 inches		1.442 inches		.889 inches
VERMILION COUNTY							
3	1	66.2		42.5		25.0	
1	¾	5.0	71.2	8.0	50.5	6.3	31.3
¾	½	7.2	78.4	11.8	62.3	12.5	43.8
½	¼	4.0	82.4	6.9	69.2	8.7	52.5
¼	⅛	4.0	86.4	6.8	76.0	12.5	65.0
⅛	0	5.0	91.4	10.9	86.9	16.3	81.3
0		8.6	100.0	13.1	100.0	18.7	100.0
Total		100.0		100.0		100.0	
Average Diameter ...			1.458 inches		1.074 inches		.753 inches
WILLIAMSON COUNTY							
3	1	94.0		70.2		60.6	
1	¾	1.6	95.6	5.7	75.9	9.1	69.7
¾	½	1.8	97.4	6.6	82.5	8.3	78.0
½	¼	0.7	98.1	3.1	85.6	2.7	80.7
¼	⅛	0.5	98.6	3.2	88.8	3.7	84.4
⅛	0	0.5	99.1	4.6	93.4	5.5	89.9
0		0.9	100.0	6.6	100.0	10.1	100.0
Total		100.0		100.0		100.0	
Average Diameter ...			1.910 inches		1.532 inches		1.383 inches

extent of disintegration was not materially different for the coals stored in the exposed bins and for those stored in the covered bins, and since during the last years of storage the covered bins were almost as much exposed to rain and weather conditions as the uncovered bins, the results of sizing tests on coals stored in covered bins are omitted. Those for the coals stored in open bins are presented in Tables 14 and 15.

TABLE 15
RESULTS OF SIZING TESTS ON SCREENINGS IN OPEN BINS

Round-Hole Screen		Original Sizes		In Storage for 1½ Years		In Storage for 6 Years	
Through Inches	Over 1 Inch	Per Cent	Cumulative Per Cent	Per Cent	Cumulative Per Cent	Per Cent	Cumulative Per Cent
SANGAMON COUNTY							
1¾	1.0	38.8		15.1		10.0	
1.0	¾	7.9	46.7	9.3	24.4	8.2	18.2
¾	½	13.2	59.9	15.6	40.0	14.5	38.7
½	¾	6.6	66.5	7.7	47.7	10.9	43.6
¾	½	7.2	73.7	9.4	57.1	15.4	59.0
¼	¼	11.2	84.9	17.2	74.3	24.5	83.5
⅛	0	15.1	100.0	25.7	100.0	16.5	100.0
Total		100.0		100.0		100.0	
Average Diameter ..			.768 inches		.498 inches		.452 inches
VERMILION COUNTY							
1¾	1	19.0		11.3		8.8	
1.0	¾	8.9	27.9	6.3	17.6	6.3	15.1
¾	½	14.8	42.7	12.9	30.5	12.1	27.2
½	¾	8.5	51.2	9.3	39.8	9.3	36.5
¾	½	11.1	62.3	11.8	51.6	13.9	50.4
¼	¼	16.4	78.7	21.0	72.6	29.0	79.4
⅛	0	21.3	100.0	27.4	100.0	20.6	100.0
Total		100.0		100.0		100.0	
Average Diameter ..			.548 inches		.425 inches		.403 inches
WILLIAMSON COUNTY							
1¾	1	18.9		19.0		6.0	
1.0	¾	9.0	27.9	9.2	28.2	5.5	18.2
¾	½	14.4	42.3	15.4	43.6	11.5	32.7
½	¾	8.5	50.8	9.5	53.1	9.3	43.6
¾	½	10.4	61.2	10.6	63.7	14.4	59.0
¼	¼	15.4	76.6	17.0	80.7	32.2	83.5
⅛	0	23.4	100.0	19.3	100.0	21.1	100.0
Total		100.0		100.0		100.0	
Average Diameter ..			.542 inches		.557 inches		.255 inches

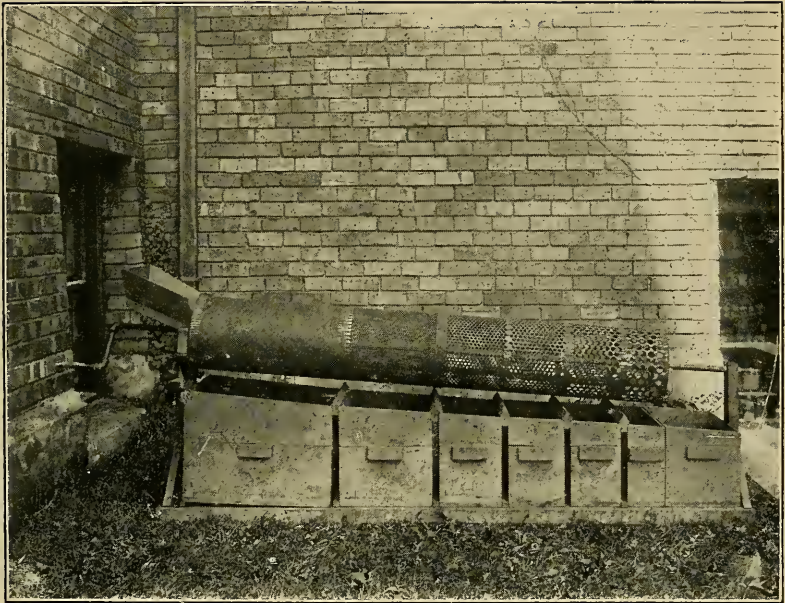


FIG. 3. REVOLVING SCREENS USED IN SIZING TESTS
(Reproduced from Univ. of Ill. Eng. Exp. Sta. Bul. 78, 1909)

A condensed summary of the detailed results presented in Tables 14 and 15 is given in Table 16. The total dust passing through a $\frac{1}{4}$ -inch screen is taken as the factor indicating the increase of fine material, and percentages of increase for the two periods are based

TABLE 16
INCREASE IN FINE MATERIAL AFTER ONE AND ONE-HALF AND SIX YEARS
(BASIS OF REFERENCE, THE TOTAL COARSE MATERIAL IN THE
ORIGINAL COAL PASSING OVER $\frac{1}{4}$ -INCH SCREEN)

Table No.	Coal and County	How Stored	Initial Storage	After 1½ Years		After 6 Years	
			Dust Passing $\frac{1}{4}$ -Inch Screen	Dust Passing $\frac{1}{4}$ -Inch Screen	Percentage Increase of Fine Material Referred to Original Coal over $\frac{1}{4}$ -Inch	Dust Passing $\frac{1}{4}$ -Inch Screen	Percentage Increase of Fine Material Referred to Original Coal over $\frac{1}{4}$ -Inch
1	Nut—Sangamon.....	Open	1.2	13.2	12.1	31.9	31.0
2	Nut—Vermilion.....	Open	13.6	24.0	12.0	35.0	24.7
3	Nut—Williamson....	Covered	1.4	11.0	11.1	13.9	12.6
4	Screenings—Sangamon.....	Covered	26.3	38.5	16.5	45.1	25.5
5	Screenings—Vermilion.....	Open	37.7	48.4	17.1	49.6	19.1
6	Screenings—Williamson....	Covered	38.8	45.4	10.7	50.6	19.2

on the total amount of material in the original coal over $\frac{1}{4}$ -inch in size. Moreover, the samples taken from this table vary with reference to the method of storage, three being from exposed and three from covered bins. These particular samples are used in the table for the reason that the same storage lots were selected for making the standard boiler tests which are hereinafter discussed.

An examination of Table 16 shows that the rate of disintegration is consistent with the variety of coal as already discussed in connection with the absorptive capacity for oxygen, and suggests that the process of oxidation of the organic material may be quite as largely responsible for this breaking down of the particles as the oxidation of the finely divided pyrite sulphur. The latter may or may not be distributed throughout the texture of the coal; this characteristic, for Illinois coal at least, has not been determined.

VII. BOILER TESTS ON WEATHERED COAL

Upon the completion of the storage experiments at the end of the six-year period, it was decided to conduct if possible a series of boiler tests under standard conditions and to compare the results of tests with stored coal with those of similar tests in which fresh coal is used. Such an opportunity presented itself in connection with a series of boiler tests being conducted at that time by Mr. A. P. Kratz, who was making a general study of boiler losses.*

The coal which Mr. Kratz was using as standard material consisted of screenings from the Mission Field, Vermilion county, Illinois. Nineteen tests were conducted with this coal. Five additional tests were made on samples of the weathered coal, one sample each of nut coal being selected from the lots from Sangamon, Vermilion and Williamson counties, and one sample each of screenings from the Sangamon and the Williamson county lots. With reference to the ash content and the percentage of finely divided material, the properties of the stored coals did not differ greatly from those of the fresh coals. An excellent basis was, therefore, provided for comparing the efficiency of the stored and fresh coals. As might be expected, some experience had to be acquired as to the best method of handling and firing the weathered coals, and although the first test was unsatisfactory, subsequent experiments yielded results which proved trustworthy and useful as a basis of comparison.

"On the first test the coal banked slightly at the water-back, and the whole amount on the grate became clinkered. It immediately became evident that in order to run at all, the coal had to be kept away from the water-back. After the clinker had been removed a fresh start was made, and care was taken to keep the fuel bed from four to six inches away from the water-back. When this was done no further trouble was experienced."†

With reference to the relative drafts required for a given rate of combustion with the fresh and the weathered coal, the tests showed that a stronger draft is required for the weathered coal. Moreover, by comparing draft requirements for those lots which were in close agreement as to their dust content, it seems evident that the higher draft requirement is not necessarily due to a higher dust factor. The

*A. P. Kratz, "Study of Boiler Losses," Univ. of Ill. Eng. Exp. Sta. Bul. 78, 1915.

†Univ. of Ill. Eng. Exp. Sta. Bul. 78, p. 50, 1915.

explanation for this is simple if we recall the discussion already presented concerning the avidity for oxygen of freshly mined coal. This avidity for oxygen seems to be directly related to the free burning character of the coal. Pillar coal and coal that has been long in storage does not burn so freely as fresh coal. On account of this characteristic such coal is thought to have lost a large part of its heat, when as a matter of fact it may have the same number of heat units but a very different rate of combustion. Another factor, and probably a minor one, is the loss of combustible gases. Mention has been made of the fact that part of the deterioration which occurs in the first few days or weeks after mining is due to the escape of certain light volatile or gaseous fuel constituents. It is evident, therefore, that either on account of the avidity of the fresh unsaturated coal for oxygen or the presence of light fuel constituent, or because of both conditions combined, the fresh unweathered coal burns freely and gives up its heat units readily, whereas the opposite is true with weathered coal. This difference, therefore, must be offset by a stronger draft or by some combination of conditions which will effect a speeding up of the burning or oxidation process. If by this means the rate of combustion can be made to approach that of the fresh coal, a corresponding degree of efficiency should result. The correctness of this theory is borne out by the results shown in Table 17. It is to be noted that the over-all efficiency of

TABLE 17

RESULTS OF BOILER TESTS WITH MISSION FIELD FRESH COAL AND WITH WEATHERED COAL AFTER SIX YEARS IN STORAGE

Mission Field			Weathered Coal				
Test No.	Boiler H. P. Developed	Efficiency of Boiler, Furnace, and Grate Per Cent	Test No.	No.	Coal and County (Tables 5 to 10)	Boiler H. P. Developed	Efficiency of Boiler, Furnace, and Grate Per Cent
10	554.0	63.96	20	1	Nut—		
11	569.6	61.21	21	4	Sangamon.....	568.5	64.50
12	572.7	60.67	22	3	Screenings—		
13	589.0	69.87	23	6	Sangamon.....	557.2	63.05
14	555.9	64.75	24	2	Nut—		
15	506.6	65.50			Williamson.....	727.1	65.98
16	644.0	60.84			Screenings—		
					Williamson.....	509.6	60.04
					Nut—		
					Vermillion.....	655.0	64.20

the weathered coal averages quite as high as that of the fresh screenings.

The general summary covering the behavior of the coal in steam generation after six years of storage, as set forth in Bulletin 78 of the University of Illinois Engineering Experiment Station, is as follows:

“1. Burning weathered coal is largely a question of correct handling and ignition. Under these circumstances it gives as good results as fresh screenings.

“2. Weathered coal requires a little thinner fire and more draft than fresh screenings.

“3. When using weathered coal the fuel bed should not approach any nearer to the water-back than from 4 to 6 in., otherwise trouble with clinker is experienced.

“4. Practically as high capacity was obtained with weathered coal as with the other coals used, and, if anything, the fuel bed requires less attention.”

In this reference, attention is called further to the fact that the results obtained and the conclusions presented are based on the heat values in the coal as fired and do not take into account the matter of deterioration. But it has already been shown in the previous discussion that the deterioration is largely apparent in a physical change and that the actual loss of heat value is small. Hence, the efficiency factors developed in the tests may be accepted as fairly representing results obtainable on weathered coal in which the heat loss resulting from weathering is practically negligible.

VIII. CONCLUSIONS

The facts presented in the preceding sections of this bulletin justify the following conclusions:

(1) Bituminous coal can be stocked without appreciable loss of heat values provided the temperature is not allowed to rise above 180 degrees F. In fact, there is no appreciable evolution of CO_2 at temperatures below 260 degrees F.

(2) The indicated heat loss per pound of coal is due more largely to an increase in weight of a unit mass of coal resulting from the absorption of oxygen than to an actual deterioration or loss of heat units.

(3) Freshly mined coal has a large capacity for absorbing oxygen which combines chemically with both the organic combustible and the iron pyrites present.

(4) The combination of oxygen with coal at ordinary temperatures generates a small increment of heat.

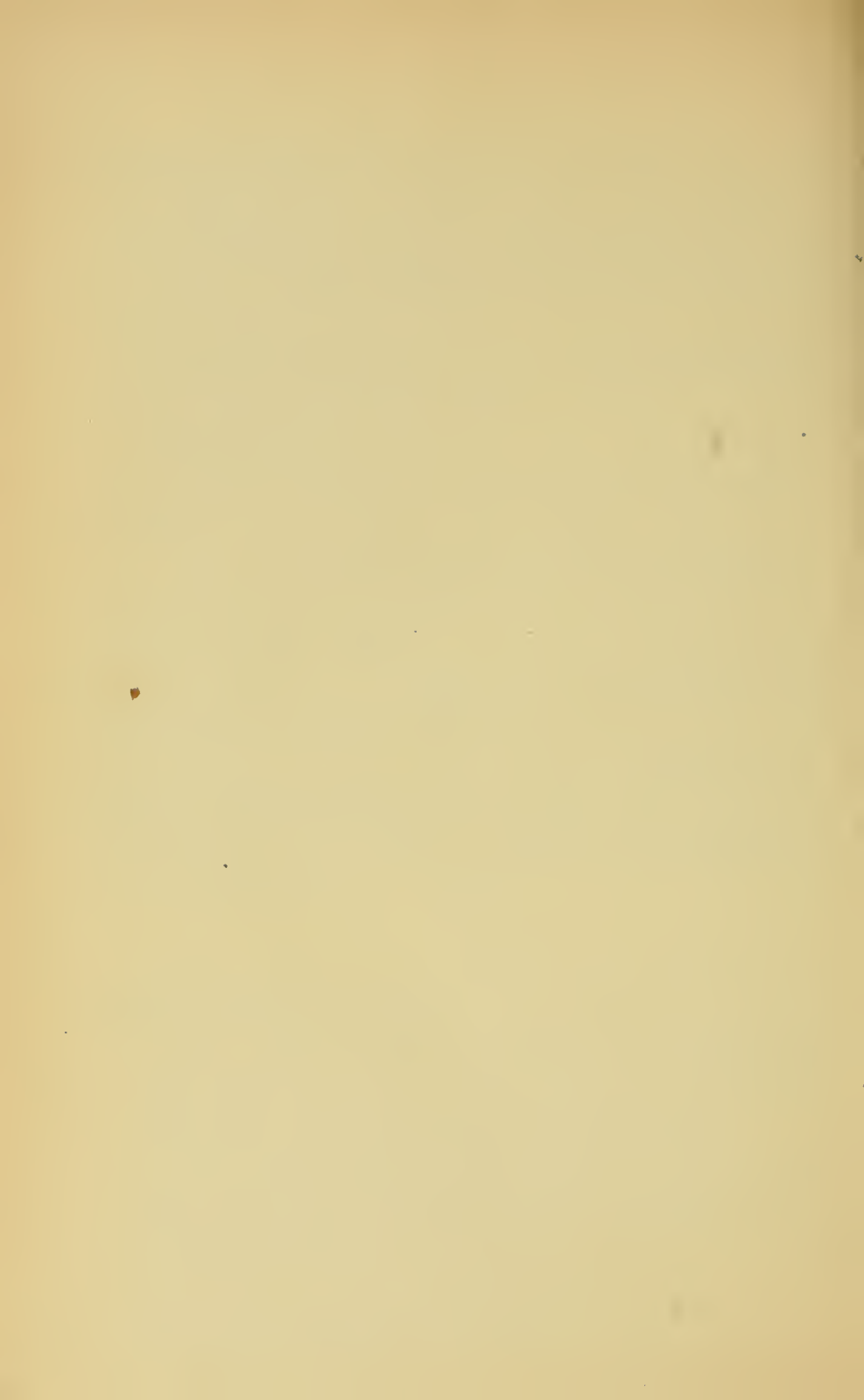
(5) The rapidity with which oxygen is absorbed depends upon the temperature of the mass and the extent of the superficial area exposed, that is, the fineness of division of the coal.

(6) If heat is generated by this slow process of oxidation more rapidly than it is lost by radiation, the acceleration of the reaction causes a rise in temperature which quickly brings the mass up to the danger point. A temperature of 180 degrees F. is named as the danger point because, if the coal reaches that temperature, practically all of the free moisture is vaporized and the further rise in temperature will be very rapid.

(7) Any method of storage to be successful must either check or prevent the absorption of oxygen to such an extent that the generation of heat shall not proceed so rapidly as to exceed natural heat losses due to radiation.

(8) Underwater storage prevents loss of heat values, and is not accompanied by deterioration in physical properties, such as slacking.

(9) Dry storage is far more safely undertaken if the fine material is screened out at the storage yard and the lump only, preferably sized, is stocked.



LIST OF PUBLICATIONS OF THE ENGINEERING EXPERIMENT STATION

- Bulletin No. 1.* Tests of Reinforced Concrete Beams, by Arthur N. Talbot, 1904. *None available.*
- Circular No. 1.* High-Speed Tool Steels, by L. P. Breckenridge. 1905. *None available.*
- Bulletin No. 2.* Tests of High-Speed Tool Steels on Cast Iron, by L. P. Breckenridge and Henry B. Dirks. 1905. *None available.*
- Circular No. 2.* Drainage of Earth Roads, by Ira O. Baker. 1906. *None available.*
- Circular No. 3.* Fuel Tests with Illinois Coal (Compiled from tests made by the Technological Branch of the U. S. G. S., at the St. Louis, Mo., Fuel Testing Plant, 1904-1907), by L. P. Breckenridge and Paul Diserens. 1909. *Thirty cents.*
- Bulletin No. 3.* The Engineering Experiment Station of the University of Illinois, by L. P. Breckenridge. 1906. *None available.*
- Bulletin No. 4.* Tests of Reinforced Concrete Beams, Series of 1905, by Arthur N. Talbot 1906. *Forty-five cents.*
- Bulletin No. 5.* Resistance of Tubes to Collapse, by Albert P. Carman and M. L. Carr. 1906. *None available.*
- Bulletin No. 6.* Holding Power of Railroad Spikes, by Roy I. Webber, 1906. *None available.*
- Bulletin No. 7.* Fuel Tests with Illinois Coals, by L. P. Breckenridge, S. W. Parr, and Henry B. Dirks. 1906. *None available.*
- Bulletin No. 8.* Tests of Concrete: I, Shear; II, Bond, by Arthur N. Talbot. 1906. *None available.*
- Bulletin No. 9.* An Extension of the Dewey Decimal System of Classification Applied to the Engineering Industries, by L. P. Breckenridge and G. A. Goodenough. 1906. Revised Edition 1912. *Fifty cents.*
- Bulletin No. 10.* Tests of Concrete and Reinforced Concrete Columns, Series of 1906, by Arthur N. Talbot. 1907. *None available.*
- Bulletin No. 11.* The Effect of Scale on the Transmission of Heat through Locomotive Boiler Tubes, by Edward C. Schmidt and John M. Snodgrass. 1907. *None available.*
- Bulletin No. 12.* Tests of Reinforced Concrete T-Beams, Series of 1906, by Arthur N. Talbot. 1907. *None available.*
- Bulletin No. 13.* An Extension of the Dewey Decimal System of Classification Applied to Architecture and Building, by N. Clifford Ricker. 1907. *None available.*
- Bulletin No. 14.* Tests of Reinforced Concrete Beams, Series of 1906, by Arthur N. Talbot. 1907. *None available.*
- Bulletin No. 15.* How to Burn Illinois Coal Without Smoke, by L. P. Breckenridge. 1908. *None available.*
- Bulletin No. 16.* A Study of Roof Trusses, by N. Clifford Ricker. 1908. *None available.*
- Bulletin No. 17.* The Weathering of Coal, by S. W. Parr, N. D. Hamilton, and W. F. Wheeler. 1908. *None available.*
- Bulletin No. 18.* The Strength of Chain Links, by G. A. Goodenough and L. E. Moore. 1908. *Forty cents.*
- Bulletin No. 19.* Comparative Tests of Carbon, Metallized Carbon and Tantalum Filament Lamps, by T. H. Amrine. 1908. *None available.*
- Bulletin No. 20.* Tests of Concrete and Reinforced Concrete Columns, Series of 1907, by Arthur N. Talbot. 1908. *None available.*
- Bulletin No. 21.* Tests of a Liquid Air Plant, by C. S. Hudson and C. M. Garland. 1908. *Fifteen cents.*
- Bulletin No. 22.* Tests of Cast-Iron and Reinforced Concrete Culvert Pipe, by Arthur N. Talbot. 1908. *None available.*
- Bulletin No. 23.* Voids, Settlement, and Weight of Crushed Stone, by Ira O. Baker. 1908. *Fifteen cents.*
- *Bulletin No. 24.* The Modification of Illinois Coal by Low Temperature Distillation, by S. W. Parr and C. K. Francis. 1908. *Thirty cents.*
- Bulletin No. 25.* Lighting Country Homes by Private Electric Plants, by T. H. Amrine. 1908. *Twenty cents.*

*A limited number of copies of bulletins starred is available for free distribution.

Bulletin No. 26. High Steam-Pressures in Locomotive Service. A Review of a Report to the Carnegie Institution of Washington, by W. F. M. Goss. 1908. *Twenty-five cents.*

Bulletin No. 27. Tests of Brick Columns and Terra Cotta Block Columns, by Arthur N. Talbot and Duff A. Abrams. 1909. *Twenty-five cents.*

Bulletin No. 28. A Test of Three Reinforced Concrete Beams, by Arthur N. Talbot. 1909. *Fifteen cents.*

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**Bulletin No. 30.* On the Rate of Formation of Carbon Monoxide in Gas Producers, by J. K. Clement, L. H. Adams, and C. N. Haskins. 1909. *Twenty-five cents.*

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**Bulletin No. 38.* The Weathering of Coal, by S. W. Parr and W. F. Wheeler. 1909. *Twenty-five cents.*

**Bulletin No. 39.* Tests of Washed Grades of Illinois Coal, by C. S. McGovney. 1909. *Seventy-five cents.*

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UNIVERSITY OF ILLINOIS BULLETIN

ISSUED WEEKLY

Vol. XIV

JUNE 4, 1917

No. 40

[Entered as second-class matter Dec. 11, 1912, at the Post Office at Urbana, Ill., under the Act of Aug. 24, 1912.]

TESTS OF OXYACETYLENE WELDED JOINTS IN STEEL PLATES

BY

HERBERT F. MOORE



BULLETIN No. 98

ENGINEERING EXPERIMENT STATION

PUBLISHED BY THE UNIVERSITY OF ILLINOIS, URBANA

PRICE: TEN CENTS

EUROPEAN AGENT

CHAPMAN & HALL, LTD., LONDON

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ENGINEERING EXPERIMENT STATION

BULLETIN No. 98

JUNE, 1917

TESTS OF OXYACETYLENE WELDED
JOINTS IN STEEL PLATES

BY

HERBERT F. MOORE

RESEARCH PROFESSOR OF ENGINEERING MATERIALS

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TESTS OF OXYACETYLENE WELDED JOINTS IN STEEL PLATES

1. *Introduction.*—This bulletin gives the results of a series of tests of the strength of oxyacetylene welded joints in mild steel plates. The joints were welded by skilled workmen in a plant especially equipped for oxyacetylene welding.

Bulletin No. 45 of the University of Illinois Engineering Experiment Station, "The Strength of Oxyacetylene Welds in Steel," by Herbert L. Whittemore, gives the results of tests of strength of welds made under repair shop conditions; it also gives a detailed discussion of the technique of welding with the oxyacetylene blow torch.

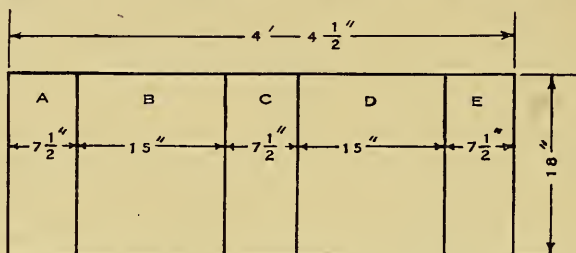
2. *Acknowledgment.*—The plates were furnished by the Oxweld Acetylene Company, and the welding was done by them at their Chicago plant. The tests of static strength of welds were made by Messrs. G. W. Watts and E. A. Brown of the class of 1915 of the College of Engineering of the University of Illinois. Messrs. Brown and Watts also acted as inspectors during the welding of the test plates. Director B. W. Benedict of the University of Illinois shop laboratories coöperated with the writer in the general planning of the tests. The tests were all made in the laboratory of applied mechanics of the University of Illinois.

3. *Tests and Test Pieces.*—Tests were made under three conditions of loading: (a) static load in tension (in a testing machine), (b) repeated load (bending), and (c) impact in tension (in a drop testing machine).

The static tension tests give an index of the resistance of the welded joint to loads applied only a few times and without heavy impact, such as floor loads in warehouses and the dead loads on bridges. The repeated stress tests give an indication of the resisting power of the welded joint to loads repeatedly applied, such as loads carried by springs and carriage axles. The impact tests give an index of the ability of the welded joints to resist sudden heavy shock without complete rupture. High resistance to rupture under impact

represents insurance against the sudden and complete failure of a part subjected to severe bending or stretching, rather than its stress-carrying ability. High resistance to rupture under impact is of importance in material for machine parts or for railway service.

PLATES OF SOFT STEEL, 4 FT. 4 $\frac{1}{2}$ IN. BY 18 IN.



PLATES WERE CUT INTO PANELS A, B, C, D, & E

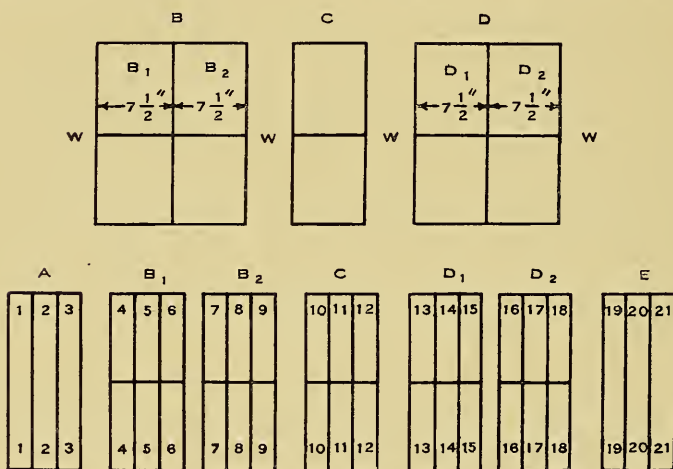


FIG. 1. PLAN OF WELDING AND CUTTING TEST PLATES AND PANELS

The plates in which the test joints were made were of steel with a carbon content of about 0.16 per cent. The following thicknesses of plate were used: No. 10 gauge, $\frac{1}{4}$ in., $\frac{1}{2}$ in., $\frac{3}{4}$ in., and 1 in. In all, eleven plates were furnished, three $\frac{1}{2}$ -in. plates, and two each of No. 10 gauge, $\frac{1}{4}$ -in., $\frac{3}{4}$ -in., and 1-in. Fig. 1 shows the plan of welding and cutting a test plate into test panels for varying heat treatment,

and also the plan of cutting these test panels into test strips to form individual test pieces. The welding of the test panels was done in the presence of inspectors from the University of Illinois, and these inspectors stamped each test strip with an identifying mark before it was cut from the test panel. The approximate speeds of welding for the various thicknesses of plate were as follows: No. 10 gauge,

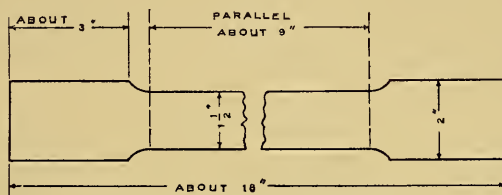


FIG. 2. TENSION SPECIMEN

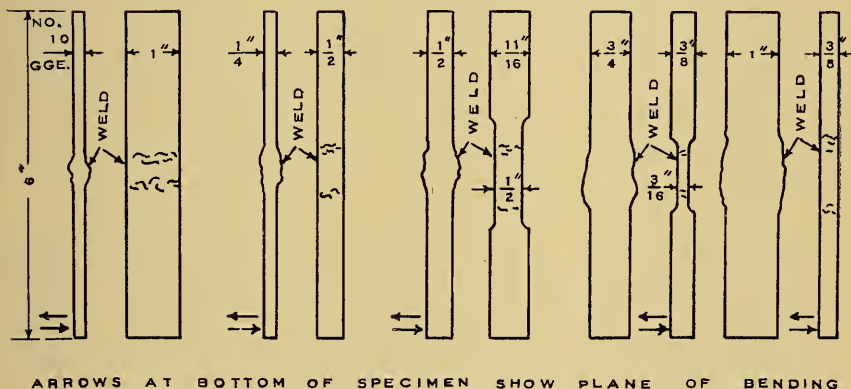


FIG. 3. SPECIMENS FOR REPEATED STRESS TESTS

0.51 in. per min.; $\frac{1}{4}$ -in. plate, 0.29 in. per min.; $\frac{1}{2}$ -in. plate, 0.22 in. per min.; $\frac{3}{4}$ -in. plate, 0.13 in. per min.; 1-in. plate, 0.11 in. per min.

The shape and size of the various test pieces cut from the test strip are shown in Figs. 2, 3, and 4. Fig. 2 shows the static tension test pieces, Fig. 3 shows the test pieces for the repeated stress tests, and Fig. 4 shows the test pieces for impact tests.

Nearly all tests were run in triplicate. There were 104 tension test specimens, 106 repeated stress specimens, and 58 impact specimens tested.

4. *Apparatus.*—The tension tests for static strength were made in a 100,000-lb. Riehle testing machine fitted with an autographic apparatus for drawing load-stretch diagrams. This apparatus was not of sufficient delicacy to permit the measurement of small elastic stretches, but did measure the comparatively large plastic stretches beyond the yield point, and did permit a good determination of the yield point to be made. The location of the yield point is plainly shown by the “knee” of the load-stretch diagram. Fig. 5 shows typical diagrams for static tests.

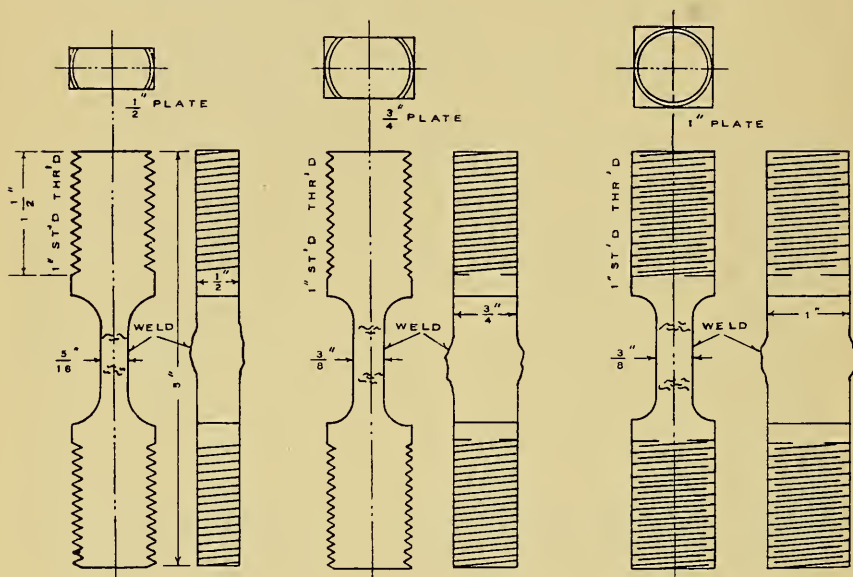


FIG. 4. SPECIMENS FOR IMPACT TESTS

The repeated stress tests were made in an Upton-Lewis endurance testing machine. Fig. 8 shows this machine. A crank *C*, with adjustable throw is driven by a motor or other source of power, and actuates a connecting rod, *D*, which, in turn, causes one end of the specimen, *Sp*, to vibrate back and forth. The specimen is held in a vise, *V*, which is pivoted at *O*; the swing of the vise round this pivot is resisted by the calibrated springs, *S*. These springs set up bending stress in the specimen. The bending moment is proportional to the width of the diagram drawn by the pencil, *R*. The pencil, *R*, draws

a diagram on a strip of paper which is moved a very short distance for each revolution of the crank, *C*. Fig. 9 shows a typical diagram from the machine. The width of each diagram is a measure of the bending stress in the specimen, and the length of each diagram is a measure of the number of repetitions of bending stress required to

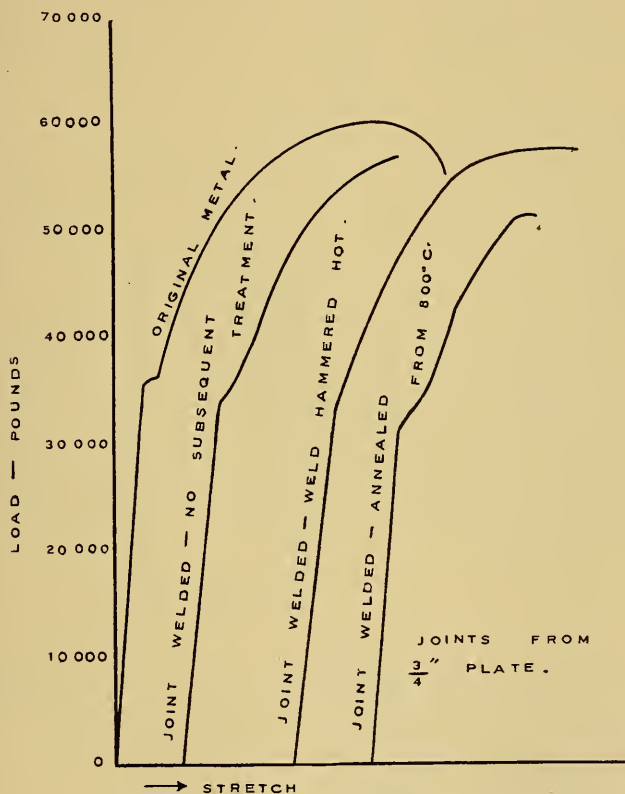


FIG. 5. TYPICAL TENSION TEST DIAGRAMS

cause failure. At *a* in Fig. 9 the specimen commenced to fail; a crack appeared in the outer fibres and progressed rapidly across the specimen. The length, *ba*, is taken as a measure of the number of repetitions required to cause failure.

Fig. 6 shows in diagram the Hatt-Turner drop testing machine used for the impact tests. The weight, *W*, is let fall from a predetermined position (shown by the broken line), and as it falls, a pencil

attached to it draws a diagram on the rotating drum, *D*. For free fall this diagram is a parabola as shown in the upper part of Fig. 7, which is a typical test diagram. In the position shown in Fig. 6, (by the solid lines), the falling weight strikes the bars, *BB*, and through them and the crosshead, *C*, tensile stress is set up in the test piece, *Sp*, sufficient to cause rupture. *O* in Fig. 7 corresponds to the location of the weight when striking the bars, *BB*, (Fig. 6) and the lower part of Fig. 7 shows the free fall after the specimen is ruptured. In rupturing the specimen kinetic energy is taken from the falling weight and its speed is reduced; after breaking the specimen another free fall takes place. Ordinates in Fig. 7 represent distance, and, since the drum revolves uniformly, abscissas represent time; hence, the slope of the diagram of Fig. 7 at any point gives a measure of the velocity of the falling weight at that point. The amount of energy absorbed in breaking the specimen can be determined if the velocities at two points in the fall, one before the weight stresses the specimen and one after rupture, are determined. In Fig. 7 let the first point be chosen at *a* and the second at *b*, and let the vertical distance from *a* to *b* be denoted by *h*. The velocity of the falling weight at *a* is given by the slope of the diagram at *a*; call this velocity V_a . Similarly determine v_b , the velocity at *b*. The kinetic energy of the falling weight is

$$\frac{1}{2} \frac{W}{g} V_a^2$$

in which *W* is the weight of the falling weight, and *g* is the acceleration due to gravity (32.2 ft. per sec. per sec.). If the weight had fallen freely to *b*, the kinetic energy at *b* would have been

$$\frac{1}{2} \frac{W}{g} V_a^2 + Wh,$$

but the velocity at *b* is actually v_b , and the kinetic energy in the falling weight at *b* is

$$\frac{1}{2} \frac{W}{g} V_b^2.$$

The energy which has been absorbed in breaking the test specimen is then

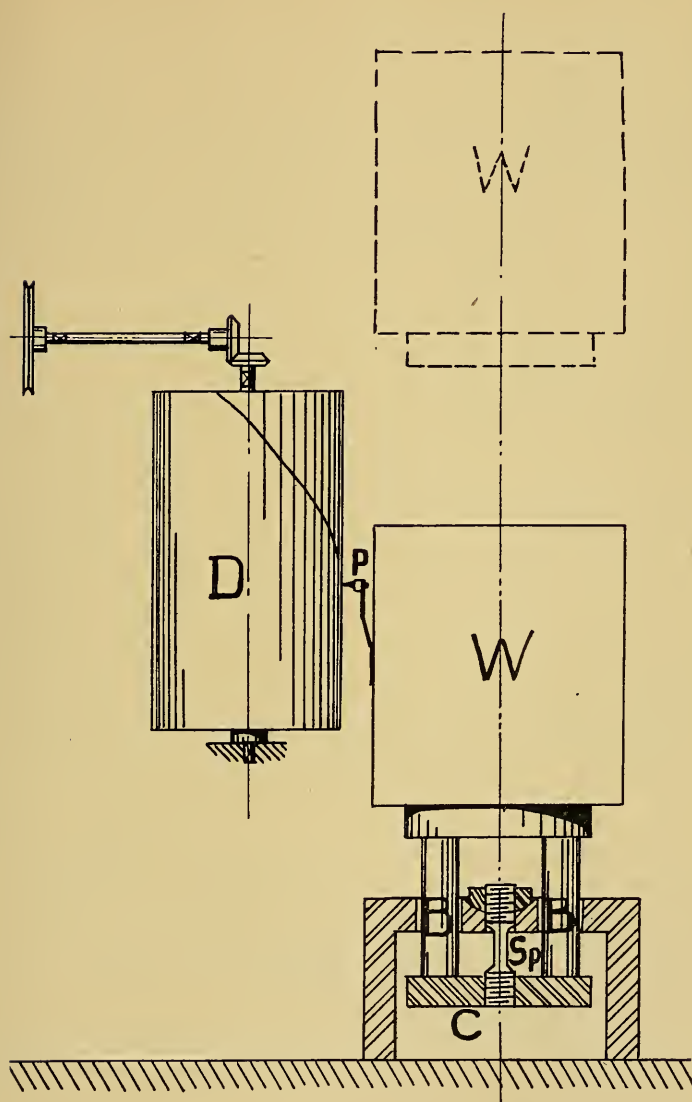


FIG. 6. DIAGRAM OF HATT-TURNER IMPACT TESTING MACHINE

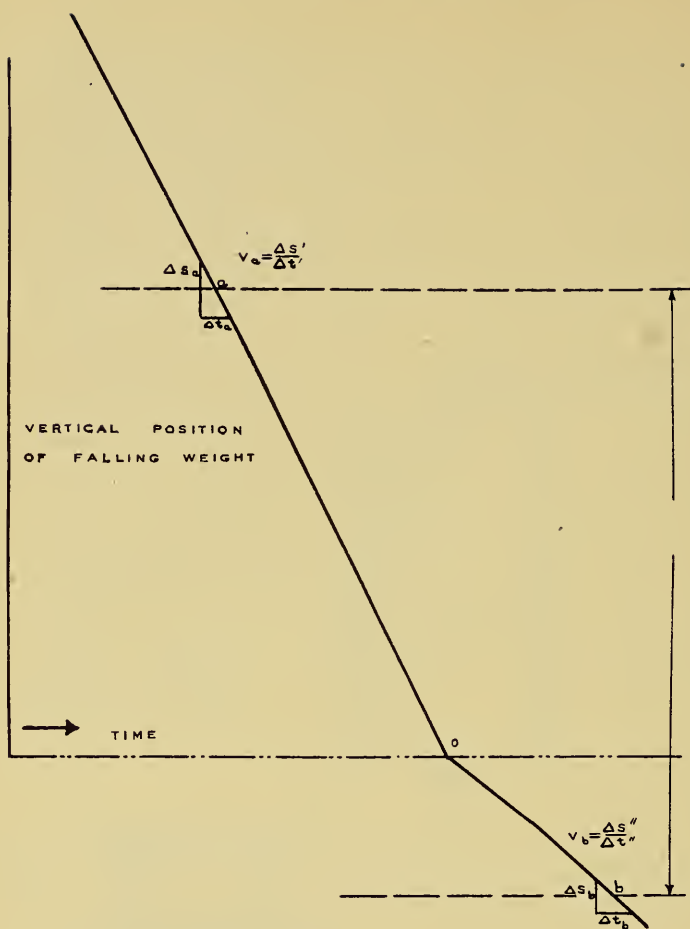


FIG. 7. TYPICAL DIAGRAM FROM HATT-TURNER IMPACT
TESTING MACHINE

$$\left(\frac{1}{2} \frac{W}{g} V_a^2 + Wh \right) - \frac{1}{2} \frac{W}{g} V_b^2.$$

In this discussion, friction of the guides for the falling weight is neglected, as is the energy absorbed in vibrations of bars, and cross-heads, but these losses are not large, and since only a comparative test between welded and unwelded specimens is desired, the described method yields results sufficiently accurate.

5. *Data and Results of Tests.*—The prime object of these tests is to furnish a comparison of the strength of oxyacetylene welds in mild steel plates with the strength of the original plate. This ratio of strength will be called the *efficiency of a joint*. The efficiency of a joint may be computed by either of two methods:

(a) The strength of a test piece containing a welded joint is compared with the strength of a test piece of equal width cut from the original plate, with no allowance made for the additional thickness of the welded test piece due to the addition of filler material, or (b) the intensity of stress at yield point or rupture for joints and for plate material may be computed from the load and the dimensions of the cross section. The efficiency given by the first method will be called the *joint efficiency* and the efficiency given by the second method will be called the *efficiency of the material* in the joint. In general, joint efficiency is of more direct practical interest than is the efficiency of the material in the joint.

In tests for static strength and for strength under repeated stress both the values for joint efficiency and for efficiency of the material are given when it is possible. (Efficiency of material cannot be given if rupture occurs outside the weld.) In the impact tests the quantity measured was *energy* (measured in inch-pounds) rather than *intensity of stress* (measured in pounds per square inch), and only the joint efficiency was determined.

In connection with the discussion of efficiency it should be noted that the breaking of a test piece outside the weld does not necessarily mean that the efficiency of the joint is 100 per cent or more. The excessive heat involved in making an oxyacetylene weld may act to weaken the material near the joint so that the strength of the original

piece is lessened and the failure will take place outside the welded joint.

The general results of the tests are given in Tables 1, 3, and 4. For the static tension tests and the impact tests, there are given for each thickness of plate: first, the results for the original material; and second, the results for the welded specimens expressed in terms of percentage of strength of the original material. The method of obtaining comparative results for the static tests and for the impact tests needs no explanation.

The method of obtaining comparative results for the repeated stress tests is somewhat complicated and will be given in detail. Table 2 gives the direct results of the repeated stress tests, the nominal stresses given in that table being the computed stress based on the thickness of the original plate. The actual stresses are based on the dimensions of the cross-section of the specimen at rupture.

It has been found* that for the range covered by these tests the relation between the computed fiber stress, S , in a specimen, and the number of repetitions, N , necessary to cause failure, may be represented with a good degree of accuracy by an equation of the form

$$S = \frac{B}{N^q} \text{ or } \log S = \log B - q \log N$$

in which B and N^q are experimentally determined constants. Plotted on logarithmic paper the graph of this equation is a straight line. The results of the repeated stress tests of oxyacetylene welded joints, when plotted on logarithmic paper, fell fairly closely along straight lines.

The test results for the specimens from each test panel were plotted on logarithmic paper and a straight line drawn to fit these results as closely as possible. The slopes of the lines for the results of all test strips were then averaged and for each test panel a line with the average slope was drawn according to the test results.

By means of this line with the average slope, the nominal stress, S , corresponding to failure at one thousand repetitions, was determined for each test panel and taken as an index of strength under

*Basquin, "The Exponential Law of Repeated Stress," Proc. A. S. T. M., 1910; Moore and Seely, "Failure of Materials under Repeated Stress," Proc. A. S. T. M., 1915; "Constants and Formulas for Repeated Stress Calculations," Proc. A. S. T. M., 1916.

TABLE 1

TENSION TESTS OF OXYACETYLENE WELDED JOINTS IN STEEL PLATES

All Values for Efficiency are given in Per Cent

I. PROPERTIES OF THE MATERIAL IN THE PLATES (SPECIMENS 1, 3, and 20)

Thickness of Plate	No. 10 Gauge	$\frac{1}{4}$ Inch	$\frac{1}{2}$ Inch	1 Inch
Yield Point (lb. per sq. in.)	30 800	38 600	33 600	33 600
Ultimate (lb. per sq. in.)	47 000	55 400	57 100	57 700
Elongation in 8 in. (per cent)	10.0	28.2	27.5	31.6

II. EFFICIENCY OF THE JOINT BASED ON YIELD POINT

Material of plate, annealed from 800 degrees C. . . .	82	87	89	90
Joint welded, no subsequent treatment	140	103	100	97
Joint welded, annealed from 800 degrees C.	107	84	92	81
Joint welded, quenched, annealed from 800 degrees C.	101	90	92	78
Joint welded, hammered while hot	128	109	101	104
Joint welded, hammered while hot, annealed from 800 degrees C.	106	80	95	94

III. EFFICIENCY OF THE MATERIAL IN THE JOINT BASED ON YIELD POINT

Material of plate, annealed from 800 degrees C. . . .	82	87	89	90
Joint welded, no subsequent treatment	...	...	81	87
Joint welded, annealed from 800 degrees C.	...	...	81	74
Joint welded, quenched, annealed from 800 degrees C.	...	...	86	76
Joint welded, hammered while hot	...	...	87	98
Joint welded, hammered while hot, annealed from 800 degrees C.	...	...	86	84

IV. EFFICIENCY OF JOINT BASED ON THE ULTIMATE

Material of plate, annealed from 800 degrees C. . . .	91	96	92	89
Joint welded, no subsequent treatment	114	101	97	79
Joint welded, annealed from 800 degrees C.	104	87	87	72
Joint welded, quenched, annealed from 800 degrees C.	103	95	83	79
Joint welded, hammered while hot	128	101	92	88
Joint welded, hammered while hot, annealed from 800 degrees C.	106	83	89	82

V. EFFICIENCY OF THE MATERIAL IN THE JOINT BASED ON THE ULTIMATE

Material of plate, annealed from 800 degrees C. . . .	91	96	92	89
Joint welded, no subsequent treatment	...	73*	77	71
Joint welded, annealed from 800 degrees C.	...	...	75	65
Joint welded, quenched, annealed from 800 degrees C.	...	...	78	76
Joint welded, hammered while hot	...	...	77	80
Joint welded, hammered while hot, annealed from 800 degrees C.	...	...	76	74

*Each value is the average result from three tests except the value starred, which is the average result of two tests.

TABLE 2

REPEATED STRESS TESTS OF OXYACETYLENE WELDED JOINTS IN STEEL PLATES

The Repeated Stress Tests were made in an Upton-Lewis Endurance Testing Machine with a Speed of 250 r.p.m.

Specimen	Thick-ness of Plate, Inches	Computed Fiber Stress Lb. per Sq. In.		Repetitions before Failure	Specimen	Thick-ness of Plate, Inches	Computed Fiber Stress Lb. per Sq. In.		Repetitions before Failure	
		Nominal	Actual				Nominal	Actual		
Material in Plate	No. 10 Gauge	45 000	44 200	13 200	Joint Welded, Annealed from 800 degrees C.	No. 10 Gauge	46 500	39 300	3 940	
		25 100	25 700	65 600			35 500	32 000	29 400	
		30 600	29 100	11 400			40 500	31 600	4 100	
		55 600	56 600	3 200			1/4	44 200	37 800	11 700
		32 400	32 600	33 800				36 600	32 200	78 500
	29 400	28 300	35 600	60 300		51 700		1 750		
	1/4	42 600	47 000	3 470		1/2	50 600	34 900	2 250	
		58 200	59 000	4 120			33 700	28 700	21 800	
		42 200	40 600	9 200			58 100	42 100	800	
	1/2	35 100	34 300	62 900		3/4	49 700	43 200	800	
		58 800	52 200	1 000			37 400	33 100	2 550	
	3/4	36 000	32 100	34 000		1	44 000	40 000	5 350	
		42 300	40 500	9 200			34 600	30 100	10 600	
		33 400	32 100	88 000			26 100	23 700	13 800	
	1	48 400	47 000	4 800	Joint Welded, quenched, Annealed from 800 degrees C.	No. 10 Gauge	26 100	21 800	157 500	
37 400		38 100	7 000	34 600			33 500	9 500		
26 100		26 100	64 700	38 300			37 700	6 300		
36 800	32 700	15 600	1/4	55 100			48 300	750		
No. 10 Gauge	29 400	28 300		15 900			32 400	24 200	13 600	
	29 000	28 200		34 800		39 100	35 800	34 000		
	32 500	35 300		3 850		46 800	46 000	6 250		
	26 200	26 200	75 500	1/2		52 400	38 700	1 550		
1/4	30 100	28 300	72 500			33 300	26 700	32 500		
	50 900	47 800	2 470			35 700	24 900	61 000		
	47 600	46 100	1 640	3/4		42 200	34 700	3 250		
	31 800	30 800	48 600			50 900	42 500	950		
1/2	45 700	40 600	2 800	1		44 100	39 700	2 850		
	29 300	29 700	25 600			36 500	32 900	6 800		
	50 800	51 300	680			25 000	23 400	19 200		
	39 700	39 300	3 770		Joint Welded, Hammered while hot	No. 10 Gauge	60 800	58 400	5 570	
3/4	33 000	31 700	88 000	48 700			44 250	11 400		
	34 400	28 700	19 400	38 400			36 300	76 500		
	57 600	48 200	700	1/4			59 300	60 200	1 100	
1	37 400	38 100	7 000				40 700	38 800	10 000	
	26 100	26 100	64 700			62 800	32 600	107 500		
	36 800	37 200	15 600	3/4		36 800	29 600	23 500		
No. 10 Gauge	51 000	48 800	4 100			48 300	43 900	2 250		
	29 400	27 800	38 300			38 100	34 600	15 800		
	32 100	28 700	156 500			27 200	25 200	45 600		
	1/4	70 000	61 800	2 730	Joint Welded, Hammered while hot, Annealed from 800 degrees C.	No. 10 Gauge	46 500	38 700	5 000	
50 800		58 500	2 790	20 800			16 500	334 000		
33 100		26 750	239 000	1/4			55 200	52 200	1 400	
52 000		48 200	13 500				32 800	25 100	84 000	
36 200	30 900	22 800	37 500				34 900	12 600		
1/2	44 000	35 500	13 200	1/2		36 400	26 800	52 000		
	67 300	48 500	700			55 300	42 300	2 000		
	34 400	28 700	19 400			42 200	32 400	7 500		
	57 600	48 200	700			3/4	42 100	36 400	6 500	
1	45 000	40 300	1 400	36 400			32 200	61500		
	38 000	34 900	7 500	1	43 700		39 800	3 550		
	27 800	25 300	47 400		34 200		32 600	11 400		
						28 500	26 900	16 900		

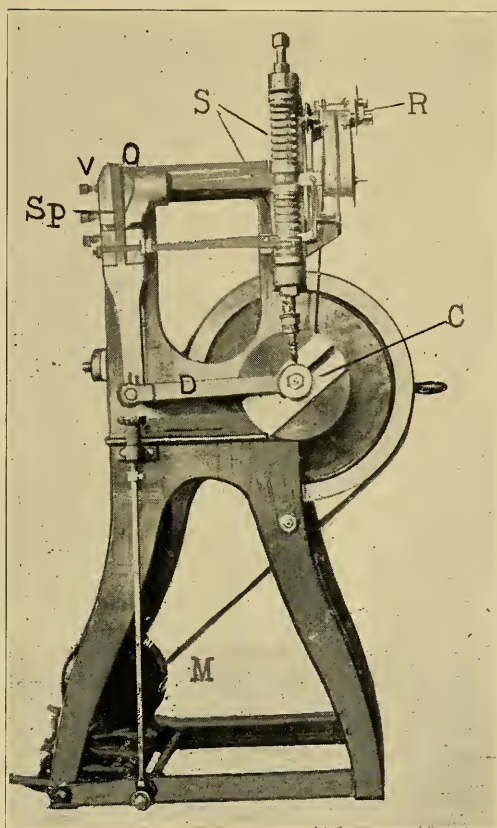


FIG. 8. UPTON-LEWIS MACHINE

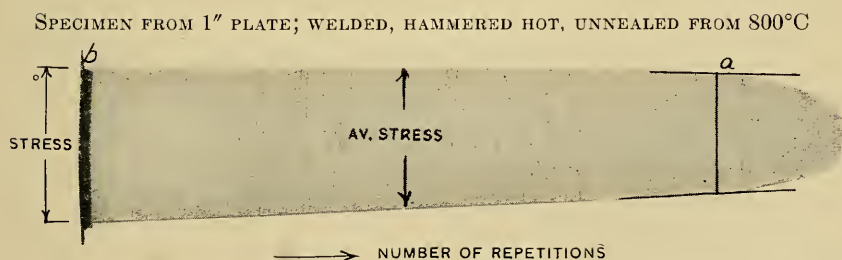


FIG. 9. TYPICAL DIAGRAM FROM UPTON-LEWIS MACHINE

TABLE 3
SUMMARY OF RESULTS OF REPEATED STRESS TESTS OF OXYACETYLENE WELDED
JOINTS IN STEEL PLATES

All Efficiencies are given in Per Cent

I. EFFICIENCY OF THE JOINTS

Thickness of plate	No. 10 gauge	$\frac{1}{4}$ inch	$\frac{1}{2}$ inch	$\frac{3}{4}$ inch	1 inch
Material in plate	100	100	100	100	100
Material in plate, annealed from 800 degrees C.	75	91	84	100	94
Joint welded, no subsequent treatment	100	125	102	88	96
Joint welded, annealed from 800 degrees C.	94	110	93	74	94
Joint welded, quenched, annealed from 800 degrees C.	90	95	94	84	86
Joint welded, hammered while hot	121	102	117	106	98
Joint welded, hammered while hot, annealed from 800 degrees C.	85	97	102	90	92

II. EFFICIENCY OF THE MATERIAL IN THE JOINTS

Thickness of plate	No. 10 gauge	$\frac{1}{4}$ inch	$\frac{1}{2}$ inch	$\frac{3}{4}$ inch	1 inch
Material in plate	100	100	100	100	100
Material in plate, annealed from 800 degrees C.	82	86	90	100	94
Joint welded, no subsequent treatment	98	108	92	76	86
Joint welded, annealed from 800 degrees C.	87	94	80	70	85
Joint welded, quenched, annealed from 800 degrees C.	84	82	81	71	79
Joint welded, hammered while hot	122	97	98	86	89
Joint welded, hammered while hot, annealed from 800 degrees C.	76	85	85	84	86

TABLE 4

RESULT OF IMPACT TENSION TESTS OF OXYACETYLENE WELDED JOINTS IN
STEEL PLATES

All Efficiencies are given in Per Cent

I. PROPERTIES OF MATERIAL IN PLATE

Thickness of plate	$\frac{1}{2}$ Inch	$\frac{3}{4}$ Inch	1 Inch
Energy required to cause rupture of specimen* (foot pounds)	511	1120†	1215

II. EFFICIENCY OF THE JOINTS

Thickness of plate	$\frac{1}{2}$ Inch	$\frac{3}{4}$ Inch	1 Inch
Material in plate, annealed from 800 degrees C.	97	93	89
Joint welded, no subsequent treatment	64	37	53
Joint welded, annealed from 800 degrees C.	66	37	35
Joint welded, quenched, annealed from 800 degrees C.	88	44	32
Joint welded, hammered while hot	89	48	58
Joint welded, hammered while hot, annealed from 800 degrees C.	72	41	53

* See Fig. 4.

† Estimated from tests of annealed joints.

repeated stress. Any number of repetitions could have been chosen as the index number; one thousand was convenient. The stress, S_p , corresponding to failure at one thousand repetitions, was determined for the plate material, and the ratio $S:S_p$ was taken as the efficiency of the test joint. If S was computed on the basis of nominal dimensions of cross-section, $S:S_p$ gives the joint efficiency under repeated stress. If S was computed on the basis of actual dimensions of cross section, $S:S_p$ gives the efficiency of the material under repeated stress.

6. *Summary.*—A few general comments on the test results are given in conclusion.

These tests were made on joints welded by skilled workmen in a shop especially fitted for oxyacetylene welding. They should not be considered as indicative of the strength of welds made in repair shops, or of welds made by workmen without special training in the use of the oxyacetylene torch.

For joints made with no subsequent treatment after welding, the joint efficiency for static tension was found to be about 100 per cent for plates $\frac{1}{2}$ in. thickness or less, and to decrease for thicker plates.

For static tension tests the efficiency of the material in the joints welded with no subsequent treatment is not greater than 75 per cent.* The joints were strengthened by working the metal after welding and were weakened by annealing at 800 degrees C.

The results of the repeated stress tests give an index of the endurance qualities of the joints, and they follow in a general way the results of the static tests.

For repeated stress tests the joint efficiency seems to be about 100 per cent for plates $\frac{1}{2}$ in. or less in thickness, while the efficiency of the material in the joint is somewhat less. Hammering or drawing the weld while hot increases the strength, and annealing from 800 degrees C. lowers it.

For static tests and for repeated stress tests, the joint efficiency sometimes reaches 100 per cent; the efficiency of the material in the joint is always less. This indicates the necessity of building up the weld to a thickness greater than that of the plate.

*Bulletin 45 of the Engineering Experiment Station of the University of Illinois, by H. L. Whittemore, showed efficiencies of material in the joints of 75-85 per cent after the operator had become proficient.

The impact tests show that oxyacetylene welded joints are decidedly weaker under shock than is the original material; for joints welded with no subsequent treatment, the strength under impact seems to be about half that of the material.

If the welded joint is worked while hot the impact-resisting qualities are slightly improved, though this does not make the joint equal to the original material in impact-resisting qualities. Annealing from 800 degrees C. seems to have very little effect on the impact-resisting qualities.

In general, the test results tend to increase confidence in the static strength and in the strength under repeated stress of carefully made oxyacetylene welded joints in mild steel plates.

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UNIVERSITY OF ILLINOIS BULLETIN

ISSUED WEEKLY

Vol. XIV

JUNE 11, 1917

No. 41

[Entered as second-class matter Dec. 11, 1912, at the Post Office at Urbana, Ill., under the Act of Aug. 24, 1912.]

THE COLLAPSE OF SHORT THIN TUBES

BY
A. P. CARMAN



BULLETIN No. 99

ENGINEERING EXPERIMENT STATION

PUBLISHED BY THE UNIVERSITY OF ILLINOIS, URBANA

PRICE: TWENTY CENTS

EUROPEAN AGENT
CHAPMAN & HALL, LTD., LONDON

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ENGINEERING EXPERIMENT STATION

BULLETIN No. 99

JUNE, 1917

THE COLLAPSE OF SHORT
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BY
A. P. CARMAN
PROFESSOR OF PHYSICS

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THE COLLAPSE OF SHORT THIN TUBES

I. RESULTS OF PREVIOUS EXPERIMENTS ON COLLAPSE OF TUBES

1. *Fairbairn's Formula.*—The problem involved in the study of the collapse of tubes is to find an equation by the application of which the collapsing pressure of a given tube can be calculated when the dimensions of the tube and the elastic properties of the material are known. The first person to attempt a solution of the problem was the noted British engineer, Sir William Fairbairn. In 1858, Fairbairn published in the Philosophical Transactions of the Royal Society of London a paper describing experiments on the collapse of about twenty-five tubes. Fairbairn's work was done at the suggestion and with the aid of the Royal Society, and the British Association for the Advancement of Science. The immediate cause was the need of such knowledge in the design of steam boilers. The tubes were "composed of a single thin iron plate bent to the required form upon a mandril and riveted and also brazed to prevent leakage into the interior." The ends were closed by means of iron disks or plugs, and the tube was placed in a large iron cylinder where it was subjected to hydraulic pressure. The interior of the tested tube was connected with the atmosphere at the upper end. The pressure was produced by a hydraulic pump and was measured by ordinary steam gauges. The diameters of the tubes collapsed were 4, 6, 8, 9, 10, and 12 inches; the lengths ranged from 19 to 60 inches; and the wall thickness was .043 of an inch for each tube. The tubes were what we describe to-day as "short" and "thin."

Fairbairn summed up the results of his experiments in the well-known formula:

$$P = 9,675,600 \frac{t^{2.19}}{ld} \quad . \quad . \quad . \quad . \quad . \quad (1)$$

"which is," he says, "the general formula for calculating the strength of wrought iron tubes subjected to external pressure within the limits indicated by the experiments, that is, provided their length is not less than 1.5 feet and not greater than 10 feet." In this formula, as in other formulas presented in this bulletin, P is collapsing pressure in pounds per square inch; t is thickness of wall in inches; l is length of tube in inches; d is diameter in inches. It is noteworthy in the history of the subject that Fairbairn's formula in some form was the

only available formula for engineers for nearly fifty years, and that his experiments seem to have been the only experiments made in this field prior to 1905 and 1906. Meanwhile, the materials and methods of construction, and the uses of tubes had changed so that Fairbairn's constants were wholly out of date.

In addition, Fairbairn's tubes, as has since been shown, were below a certain "critical length"; and, the fact of a minimum critical length not being known, the formula was generally assumed to be applicable to tubes of all lengths. Various modifications were proposed for the formula* but practically all of them were based on Fairbairn's data.

2. *Experiments of Carman.*—The first experiments on the collapse of tubes after Fairbairn's were those of A. P. Carman, which were reported and discussed in a paper read before the American Physical Society in April, 1905, and published in the *Physical Review*, Vol. XXI. These experiments were conducted with small seamless brass tubes and showed "that there is a minimum length for each tube above which the collapsing pressure is constant. Again," still quoting this paper, "we see that for lengths less than this critical minimum length, the collapsing pressure rises rapidly. As definitely as can be determined from these small tubes, the collapsing pressure varies inversely as the length, for lengths less than the critical length." It was further pointed out that Fairbairn's tubes were all less than the critical length. "These experiments on small brass tubes have shown that formulas of the Fairbairn type are inadequate, and that for tubes of sufficient length, a formula of the type proposed by Bryan and Love is more nearly true." The formula referred to is that first given by Professor G. H. Bryan† of Cambridge, England, and later deduced by other methods by A. B. Bassett‡ and A. E. H. Love§. The formula is given for long thin tubes, and is

$$P = 2E \frac{m^2}{m^2 - 1} \frac{t^3}{d^3} \dots \dots \dots (2)$$

where P is the pressure of collapse, t is the thickness of wall, d is the mean diameter, E is Young's modulus, and $\frac{l}{m}$ is Poisson's ratio.

*See Bibliography.

†Proceedings Cambridge Philosophical Society, Vol. VI.

‡Philosophical Magazine, September, 1892.

§Mathematical Theory of Elasticity, Vol. 2, p. 319, 1903.

3. *Results Obtained by Stewart, and by Carman and Carr.*—

In 1906, two papers appeared on the collapse of tubes; one by Professor R. W. Stewart* read before the American Society of Mechanical Engineers, giving the results of a large number of experiments on lap-welded steel tubes; and the other by Carman and Carr† in a bulletin of the Engineering Experiment Station of the University of Illinois, describing experiments on seamless steel, lap-welded steel, and brass tubes. All of these experiments were made on tubes longer than the “critical length,” little attention being paid to the law of lengths except to see that the tubes were longer than the critical minimum length.

The results obtained by Carman and Carr were stated as follows:

“The portion of a long tube affected by the collapse under hydraulic pressure is generally not greater than twelve times the diameter; for greater lengths the collapsing pressure is independent of the length. The law according to which the collapsing pressure varies inversely as the length, is true only for very short tubes; i. e., tubes shorter than a certain critical ‘minimum length’ which in most cases is from four to six times the diameter.

“For long thin tubes, i. e., for values of $\frac{t}{d}$ below about .025, the formula $P = k \left(\frac{t}{d}\right)^3$ is very nearly true.” The constants for brass and seamless tubes were calculated from the experimental data, and the formulas stated as follows:

a. For thin brass tubes

$$P = 25,150,000 \left(\frac{t}{d}\right)^3 \quad . \quad . \quad . \quad . \quad . \quad (3)$$

b. For thin seamless steel tubes

$$P = 50,200,000 \left(\frac{t}{d}\right)^3 \quad . \quad . \quad . \quad . \quad . \quad (4)$$

“For moderately ‘thick’ tubes, that is, for tubes having values of $\frac{t}{d}$ between .03 and .07, an equation of the form

$$P = k \frac{t}{d} - c \quad . \quad . \quad . \quad . \quad . \quad (5)$$

*Transactions of the American Society of Mechanical Engineers, 1906.

†“Resistance of Tubes to Collapse.” Univ. of Ill. Eng. Exp. Sta. Bul. 5, 1906.

was found to satisfy the experimental data, the formula becoming:

a. For brass

$$P = 93,365 \frac{t}{d} - 2474 \quad . \quad . \quad . \quad . \quad . \quad (6)$$

b. For seamless cold drawn steel

$$P = 95,520 \frac{t}{d} - 2090 \quad . \quad . \quad . \quad . \quad . \quad (7)$$

c. For lap-welded steel

$$P = 83,290 \frac{t}{d} - 1025 \quad . \quad . \quad . \quad . \quad . \quad (8)''$$

Professor Stewart found similar equations and constants from his experiments on lap-welded steel tubes.

Thus from theory and from experiments, fairly satisfactory equations for long thin tubes and for long tubes of moderate thickness have been reached. The terms "long," "thin," and moderately "thick" are used as previously defined. The equation for long, moderately thick tubes is empirical both in its constants and its form. The equation for long thin tubes is that of Bryan except that the constant is about 30 per cent less than the theoretical value, probably due, as suggested by R. V. Southwell,* to the material being beyond the elastic limits.

4. *Investigations of Southwell, and of Cook.*—Within the last three years, there has been considerable interest in the law of the collapse of "short" thin tubes, that is, in the form of the pressure-length curve near and inside the critical length. The practical interest in this part of the curve came first from the problem of spacing "collapse rings" in boiler flues. Another practical interest in the problem is found in the collapse of steel flumes by atmospheric pressure when the water is suddenly let out by accident and the internal pressure is reduced almost to zero.

The theoretical interest comes from the very important mathematical investigations of R. V. Southwell, Fellow of Trinity College, Cambridge, England, discussed in a paper entitled "On the General Theory of Elastic Stability." This paper was read before

*Philosophical Magazine, p. 504, September, 1913.

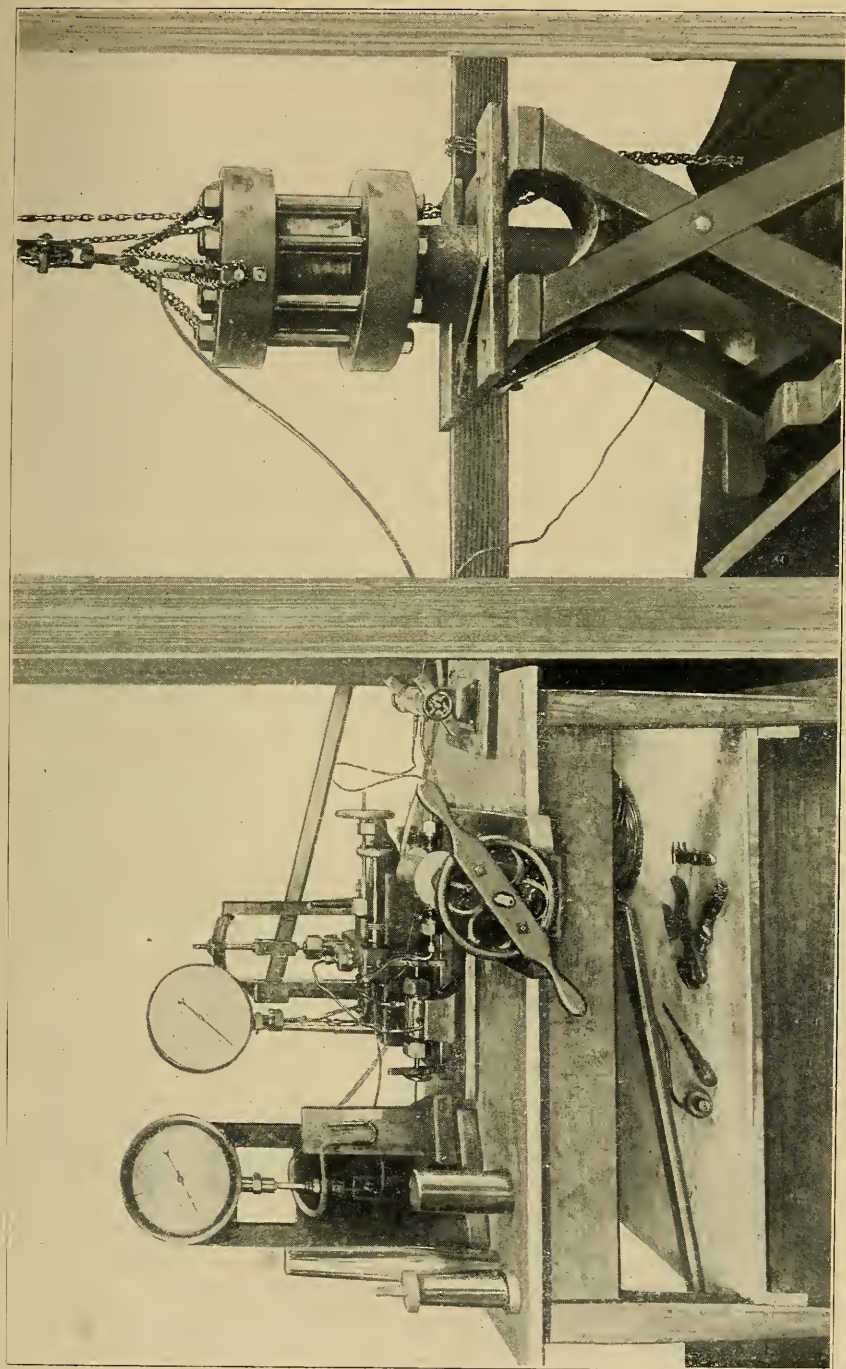


FIG. 1. APPARATUS ARRANGEMENT. PUMP AND GAUGES TO LEFT, NICKEL-STEEL CYLINDER TO RIGHT

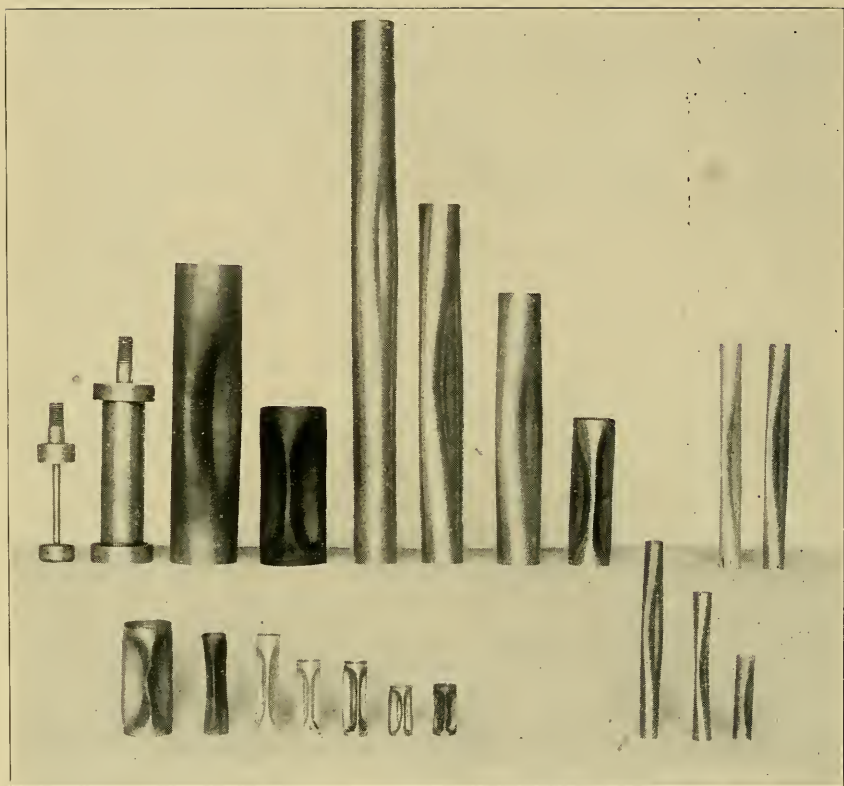


FIG. 2. TUBES THAT HAVE BEEN COLLAPSED. ON THE LEFT THE MOUNTING ARRANGEMENT IS SHOWN. LONG TUBES COLLAPSE IN TWO LOBES, SHORTER TUBES INTO THREE LOBES, AND STILL SHORTER TUBES INTO FOUR LOBES

the Royal Society of London in January, 1913.* In it Southwell developed the following formula for the collapse of thin tubes:

$$P = 2E \frac{t}{d} \left[\frac{Z}{k^4 (k^2 - 1)} \frac{d^4}{l^4} + \frac{1}{3} \frac{m^2}{m^2 - 1} (k^2 - 1) \frac{t^2}{d^2} \right] \quad (9)$$

where P is the collapsing pressure, E is Young's modulus, $\frac{l}{m}$ is Poisson's ratio, Z is a constant depending upon the end constraints, l , d ,

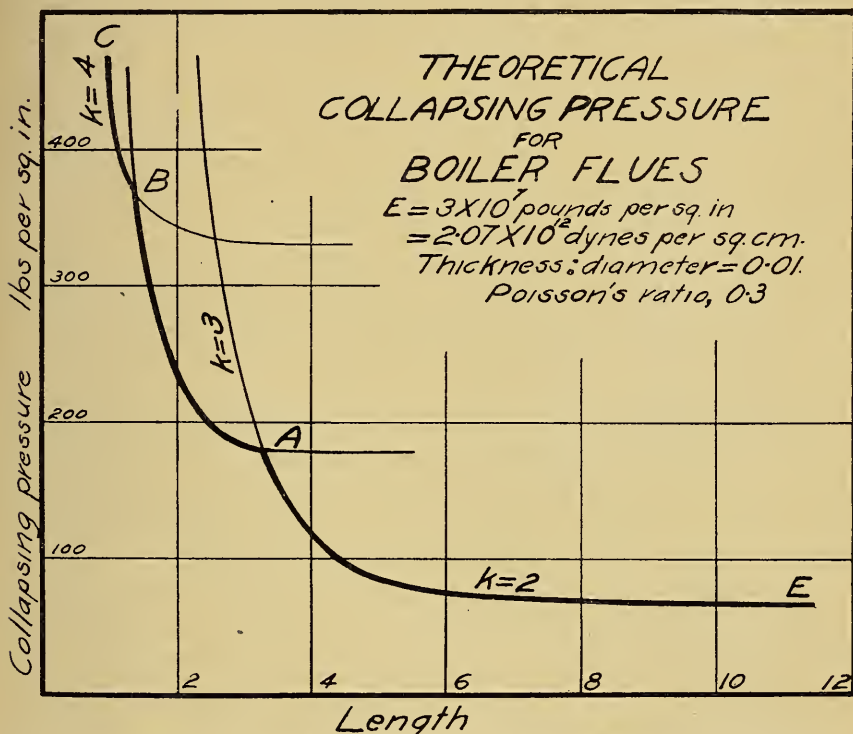


FIG. 3. CURVE OF SOUTHWELL'S EQUATION REPRODUCED FROM PHILOSOPHICAL TRANSACTIONS, 1913

and t are the length, diameter, and wall thickness, and k represents the number of lobes into which the tube collapses. It is known that long tubes collapse into two lobes, shorter tubes into three lobes, and still shorter into four lobes (See Fig. 2). This formula is represented by a family of curves, corresponding to the value of 2, 3, and 4, for k . Fig. 3 is reproduced from Southwell's paper. It shows the curves

*Philosophical Transactions of the Royal Society of London, Vol. 213A, pp. 187-244.

for two, three, and four lobes, giving the relation between collapsing pressure and length; values of Young's modulus and Poisson's ratio are assumed as noted on the curve. No simple value for Z is stated by Southwell. "From an inspection of the different curves, we see," writes Southwell, "that long tubes will always tend to collapse into the two-lobed form, since the curve for $k=2$ then gives the least value for the collapsing pressure, but that at a length corresponding to the point A , the three-lobed distortion becomes natural to the tube; and that for shorter lengths still, of which the point B gives the upper limit, the four-lobed form requires least pressure for its maintenance. Thus the true curve connecting pressure and length is the discontinuous curve $CBAE$, shown in the diagram by a thickened line." Then according to Southwell's theory, the curves which are ordinarily drawn to represent the results of experiments, such as the curves in Figs. 8 to 14, are really the envelopes of series of curves. Southwell made some experiments which seem to confirm his idea of the pressure-length curve. Our own experiments have not given any decisive proof of this feature of Southwell's formula. It is to be noted that for long tubes, that is, where $k=2$, Southwell's formula reduces to the Bryan formula.

Southwell has further discussed the theory of the collapse of tubes in three articles in the *Philosophical Magazine* for 1913 and 1915.* The third of these articles is largely a discussion of his theory with reference to some very interesting experiments on the collapse of short tubes by Gilbert Cook of the University of Manchester.†

Southwell's formula differs from the previous formula of Bryan in giving an expression for the collapsing pressures for lengths less than the "critical length." He deduces as an expression for the "critical length," $L = k \sqrt{\frac{d^3}{t}}$, " k being some constant depending upon the type of the end constraints."‡

The Southwell formula further makes the envelope of the pressure-length curve a hyperbola for lengths less than the critical length. Hence, if the critical length can be determined, the curve can be drawn and the strength of a short tube calculated. The experiments

**Philosophical Magazine*, pp. 687-698, May, 1913; pp. 502-511, September, 1913; pp. 67-77, January, 1915.

†*Philosophical Magazine*, pp. 51-56, July, 1914.

‡*Philosophical Transactions*, Vol. 213A, p. 227.

of Carman and of Stewart indicated a length of six diameters as the critical length.

Cook's experiments were made to determine the strength of short tubes and, as a part of the investigation, to determine the "critical length." He gives the results of the collapses of thirty-nine "solid drawn" steel tubes, all of three inches internal diameter, of values of $\frac{t}{d}$ varying between .0097 and .0206, and of lengths from 2.08

inches to 12.71 inches. Unfortunately Cook's apparatus limited him to tubes of less than thirteen inches in length, that is, to lengths of about four diameters, so that his curves do not reach the important bends or "critical" points. He, however, concludes from estimates that "the critical length varies from thirteen to eighteen times the diameter," and that it is given by Southwell's formula $L = k\sqrt{\frac{d^3}{t}}$,

with $k = 1.73$ for his steel tubes. Southwell in his discussion of Cook's work and the "critical length" says,* "Both theory and experiment suggest that the length of a tube sensibly affects its resistance to external pressure only in the case of comparatively short tubes, and the earliest definitions of the term 'critical length,' given almost simultaneously by Professor A. E. H. Love as 'the least length for which collapse is possible under the critical pressure,' and by A. P. Carman as a 'minimum length, beyond which the resistance of a tube to collapse is independent of the length,' were in recognition of this fact. Professor Carman concluded further from the early experiments of Fairbairn and from others which he himself conducted, that 'the collapsing pressure varies inversely as the length, for lengths less than the critical length.' That is to say, the curve suggested by him as expressing the experimental relation between collapsing pressure and length for a tube of given thickness and diameter consists of two discontinuous branches; a straight line, representing constant collapsing pressure for all lengths above the critical length, and a rectangular hyperbola intersecting this line at a point corresponding to the critical length.

"If these views were adopted, the critical length for any definite size of tube may be determined from experiments by estimating: (1) the straight line, parallel to the axis of length, which best represents the collapsing pressure for tubes of considerable length, and (2) the

*Philosophical Magazine, p. 68, January, 1915.

hyperbola which agrees best with the results for the shorter tubes; their point of intersection gives the required value. This is substantially the procedure adopted by Mr. Cook, who finds that within the range of his experiments the critical length L , thus defined, is given satisfactorily by the formula.

$$L = 1.73 \sqrt{\frac{d^3}{t}} \dots \dots \dots (10)$$

“Two of the five curves upon which Cook bases his conclusions are

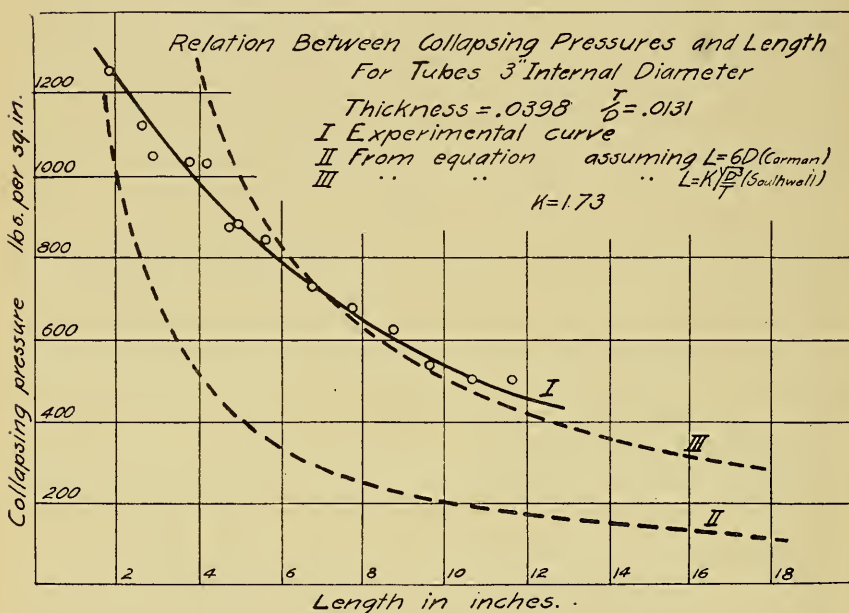


FIG. 4. PRESSURE-LENGTH CURVES SHOWING EXPERIMENTAL AND CALCULATED VALUES ACCORDING TO GILBERT COOK. REPRODUCED FROM PHILOSOPHICAL MAGAZINE, 1914

shown in Figs. 4 and 5 which are reproduced from his article. Agreement of the curve with this theory is not close, particularly for the thicker tubes. Although Cook suggests the given formula, he frankly says that ‘the tests cannot be regarded as sufficient in number or covering enough range of dimensions to confirm definitely the equation

$L = 1.73 \sqrt{\frac{d^3}{t}}$ Though there is considerable amount of data for a law on long thin tubes and for long moderately thick tubes, there are

few experimental results on the collapse of short tubes. The following experiments were undertaken in order to get more facts on the strength of short tubes of steel and of brass. To these have also been added some experiments on tubes of aluminum, glass, and hard rubber. The results obtained with the three last named materials not only have an interest in themselves, but also may have a theoretical interest in future discussions of formulas on account of the different elastic constants involved.

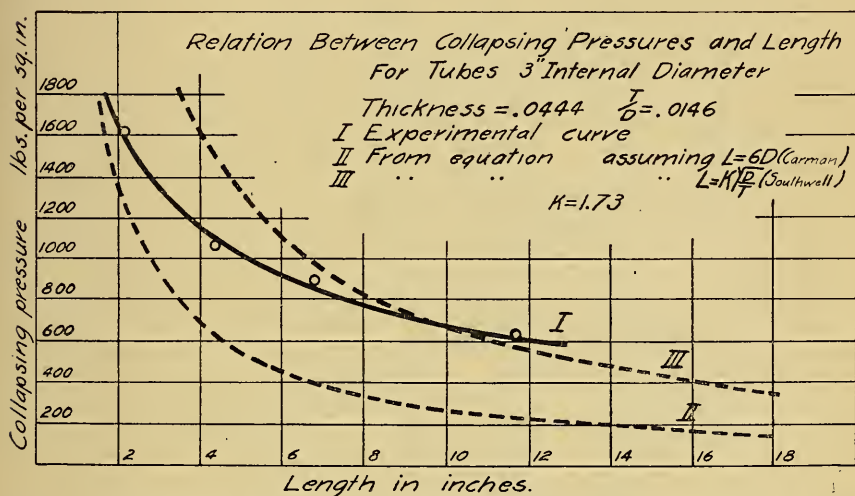


FIG. 5. PRESSURE-LENGTH CURVES SHOWING EXPERIMENTAL AND CALCULATED VALUES ACCORDING TO GILBERT COOK. REPRODUCED FROM PHILOSOPHICAL MAGAZINE, 1914

A large part of the experimental work described in this paper has been done by Stetfan F. Tanabe, M. S., Research Fellow of the Engineering Experiment Station. Credit is due him also for many of the details of the apparatus and methods.

II. METHODS AND RESULTS OF EXPERIMENTS ON THE COLLAPSE OF SHORT THIN TUBES

5. *Apparatus.*—The apparatus was for the most part that which the author used a number of years ago in previous experiments of this kind. The hydraulic pressure was produced in a stout nickel-steel tube or receptacle 40 inches in length, 5 inches in internal diam-

eter, and 7 inches in external diameter. The lower end was closed by a cast-iron plug, and the upper end by a heavy steel disk or cover which

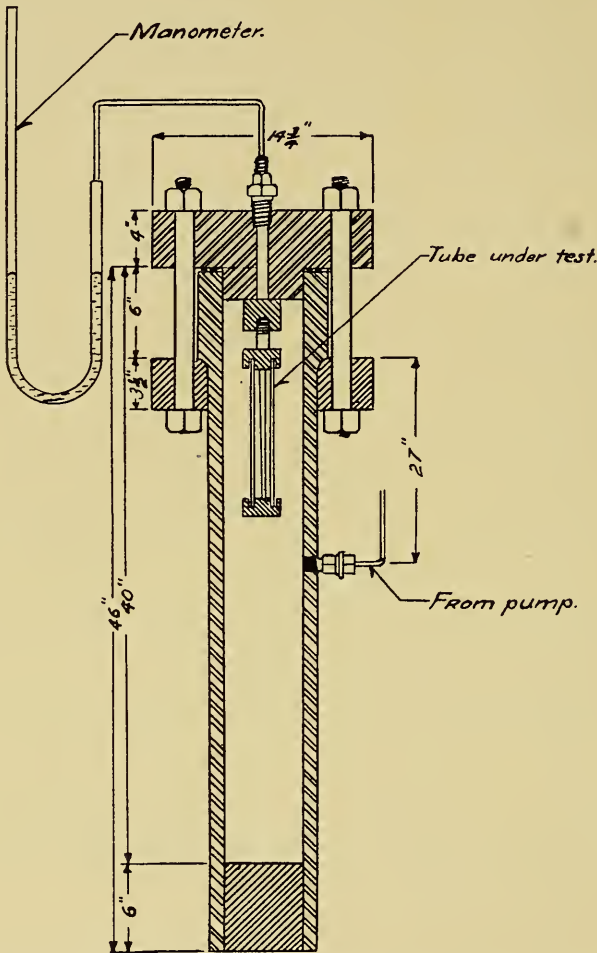


FIG. 6. SECTION OF THE NICKEL-STEEL TUBE IN WHICH THE TUBES WERE COLLAPSED BY HYDRAULIC PRESSURE

was held in place by eight circular bolts as shown in Figs. 5 and 6. A lead gasket with circular grooves in the end face of the tube prevented leakage even at the highest pressures. This tube was held in a vertical position, so that the heavy steel cover could be conveniently

handled by a chain hoist. The hydraulic pressure was produced by a Cailletet pump, made by the Société Gènevoise, and by a screw plunger, which was made in the department shop. The pressure was raised nearly to the required amount by the pump, and then brought

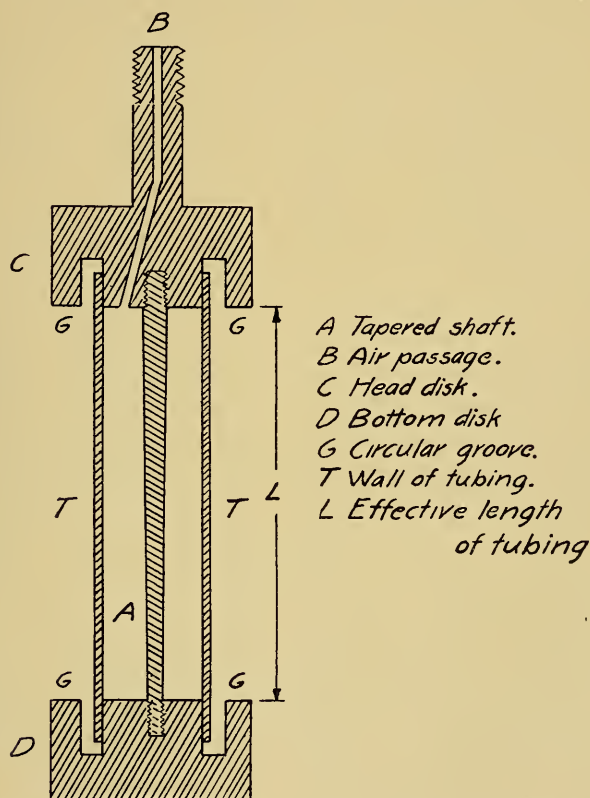


FIG. 7. CROSS-SECTION OF SPINDLE FOR MOUNTING A TUBE FOR COLLAPSE
 THE GROOVES *G* WERE FILLED WITH PACKING AND ASPHALTUM
 TO MAKE THEM WATERTIGHT

gradually to the pressure of collapse by means of the screw plunger. It was measured by three tested gauges of the Bourdon type. The arrangement for closing the tube for collapse is shown in Fig. 7. This was the result of several trials, and proved very effective and easy to use. The steel end plugs were turned so that the tube could slide over them with a close fit, but without strain. The length of the

tube thus supported on the inside at each end was about one-eighth inch for the small tubes, and as large as one half-inch for the three-inch tubes. The groove left around the tube was for the packing. The end of the tube was wrapped with "friction" tape—the rubberized tape used for insulation in electrical wiring—and the surrounding groove was packed with it. Melted asphaltum was then poured over the tape, and in some cases the whole end was coated by dipping it into the asphaltum. There was seldom any trouble with leakage even at the highest pressures. The end constraint was, by this arrangement, reduced practically to that of the support of the internal plug. To eliminate the end pressure on the tube, an internal bolt was used which was threaded at both ends and screwed down by means of tapped holes at the centers of the end plugs. As this bolt did not come through the end plugs, a serious source of leakage was avoided. The bolt or shaft was tapered so that it could be easily withdrawn from the collapsed tube. The upper plug was turned down and threaded so as to screw into a hole tapped into the large steel-disk cover of the nickel-steel receptacle. A small canal through this connected the interior of the tested tube with a manometer. The pressure on the inside was atmospheric except at the moment of collapse. The steel and brass tubes were carefully machined to the required thicknesses, so that results have been obtained for several values of $\frac{t}{d}$ for each length of tube.

Three gauges were used, the connections being such that any one or all could be used in each case. Two of the gauges were made by Shaffer and Budenberg of Magdeburg, Germany, the third by the Crosby Gage Company. The first gauge read to a maximum of 300 kilograms per square centimeter, each scale division representing 5 kilograms per square centimeter; the second, to a maximum of 1000 kilograms per square centimeter, each scale division representing 10 kilograms per square centimeter; and the third to 2000 pounds per square inch, each scale division representing 50 pounds per square inch. These gauges were tested by using a Crosby Fluid Pressure Scale. This is the standard rotating piston balance made by the Crosby Gage Company for testing gauges up to a pressure of 25,000 pounds per square inch. The readings of the gauges were corrected in accordance with these tests; the corrections were so small that they could have been neglected for these experiments.

6. *Results.*—The results of the experiments are given in Tables 1 to 7 inclusive (pages 33 to 36) and in Figs. 8 to 14 inclusive of this

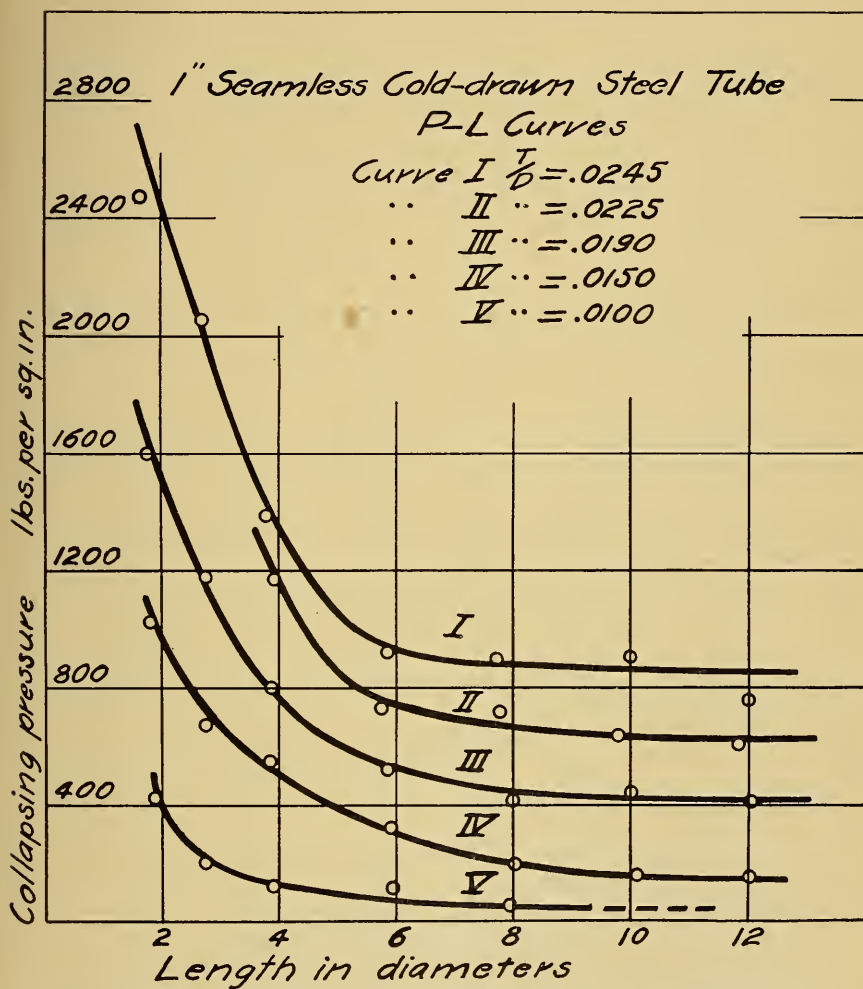


FIG. 8. PRESSURE-LENGTH CURVES FOR FIVE DIFFERENT THICKNESSES OF ONE-INCH SEAMLESS STEEL TUBES

bulletin. The number of lobes of the collapsed metal tubes is given in most cases. The number is recorded on account of its importance in the theory of stability and instability, which has been discussed by Southwell. In the collapse of glass tubes, we had the very striking

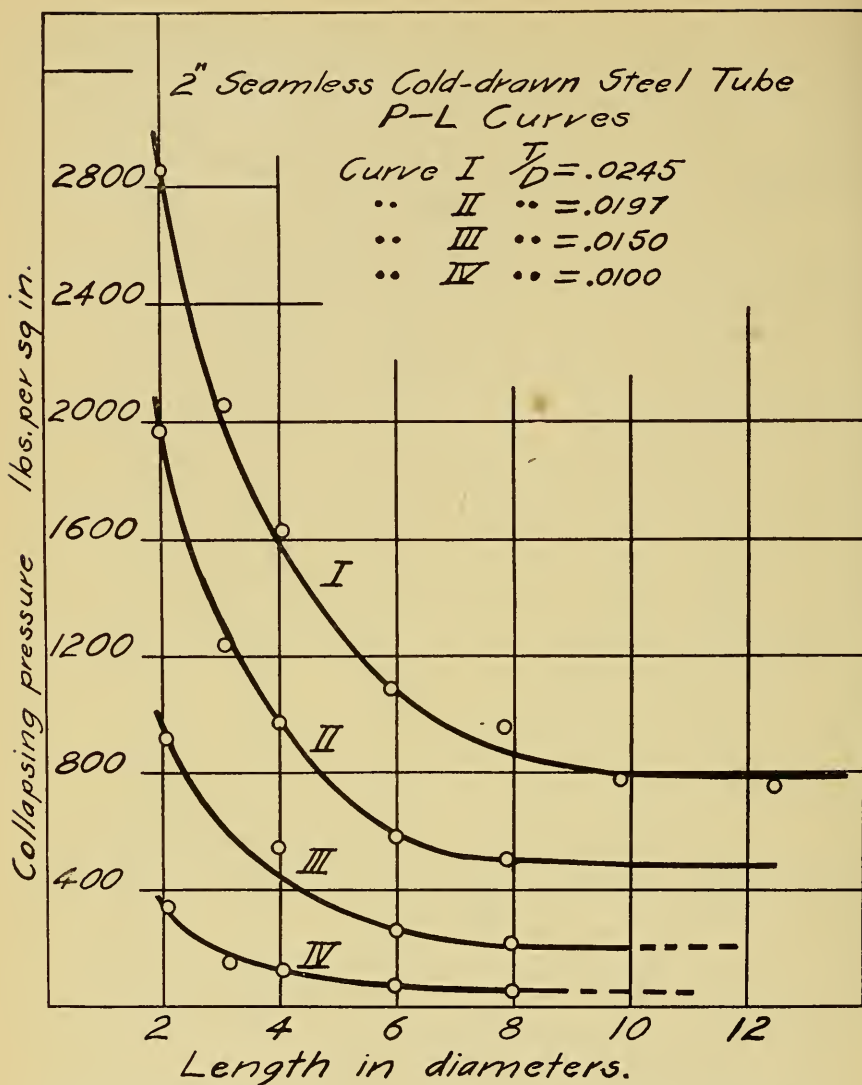


FIG. 9. PRESSURE-LENGTH CURVES FOR FOUR DIFFERENT THICKNESSES OF TWO-INCH SEAMLESS STEEL TUBES

occurrence of the glass being reduced to a fine powder. The data as given represent the results of the collapse of about one hundred and fifty tubes. In nearly all cases of the steel and the brass tubes, there

were two tubes of the given dimensions; thus an average could be made for each point. This duplication also gave an immediate check on

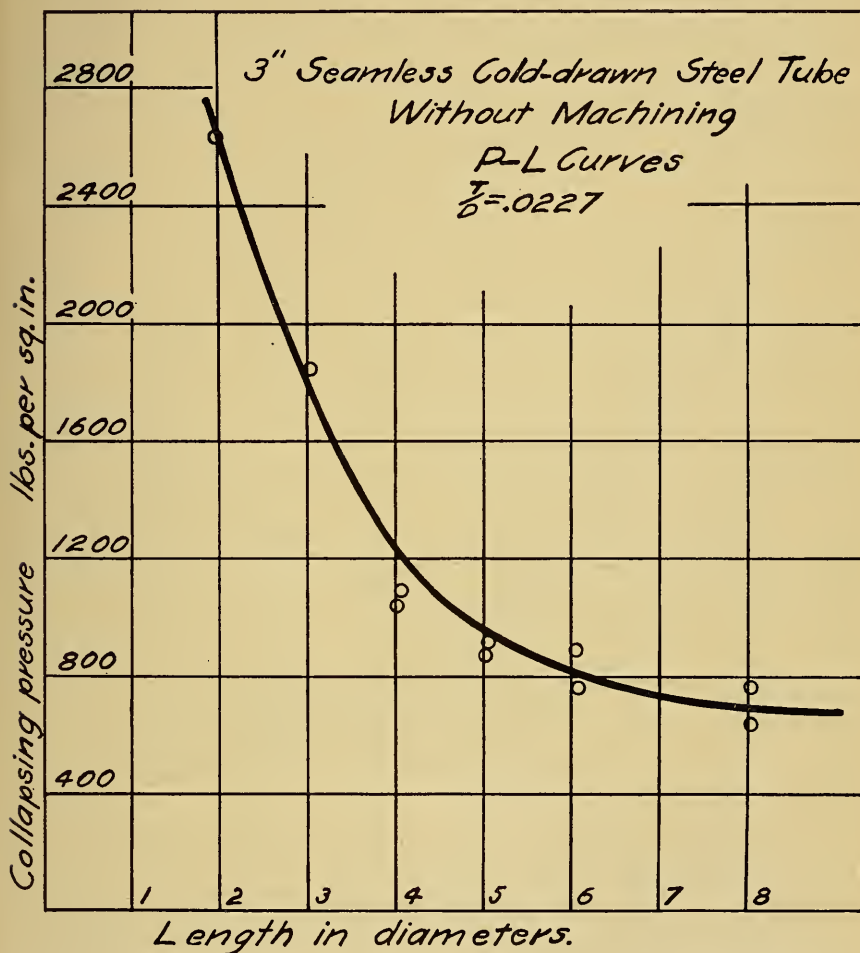


FIG. 10. PRESSURE-LENGTH CURVE FOR A THREE-INCH SEAMLESS STEEL TUBE
WITH THICKNESS OF $\frac{t}{d} = .0227$

freak collapses. Very few freak collapses occurred, however, and these could almost always be explained by the irregularities in dimensions or in material that appeared upon the inspection of the collapsed tube. While the machine work on these tubes was done with great

care by the mechanics of the department, Messrs. Hays and Buchanan, it was impossible to get the value of $\frac{t}{d}$ exactly the same each time. It was necessary therefore to make corrections in the

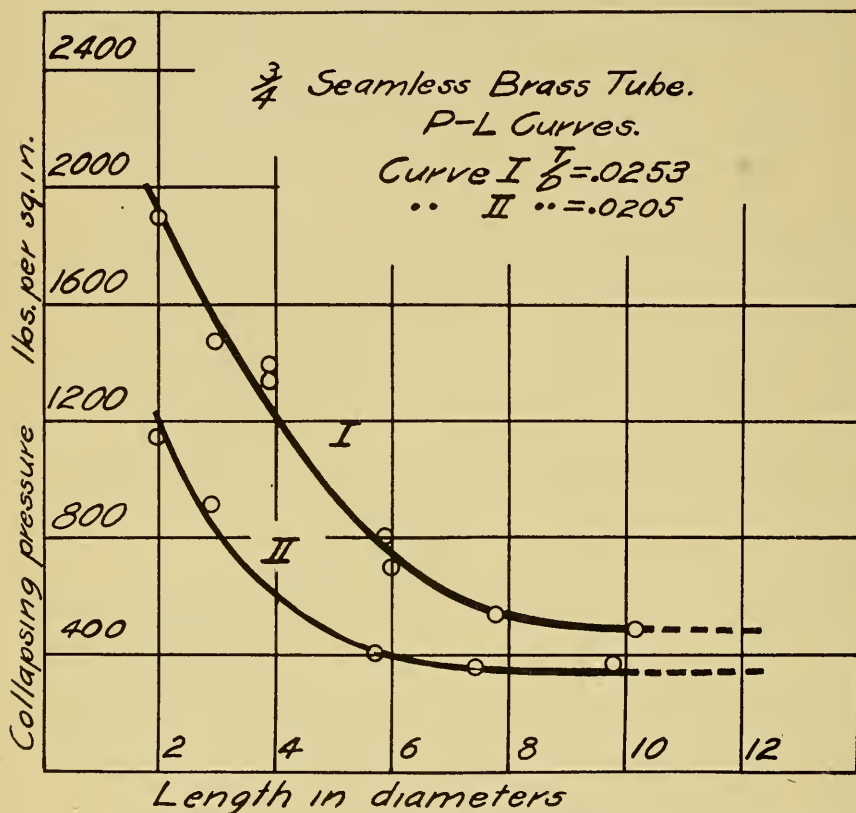


FIG. 11. PRESSURE-LENGTH CURVE FOR THREE-QUARTER-INCH BRASS TUBE WITH THICKNESS OF $\frac{t}{d} = .0205$ AND $.0253$

observed collapsing pressures so as to have sets of results for a pressure-length curve with $\frac{t}{d}$ constant. These corrections were made by interpolation; it was assumed that the collapsing pressures varied as $(\frac{t}{d})^3$ for constant length. Since the total variations of $\frac{t}{d}$ in

these cases were very small, there is little assumption in the interpolations and corrections.

Observations were also made of about fifty steel tubes of half-inch and three-quarter-inch diameters. The results were very irregular, due partly to unavoidable variations in thickness, but more to a lack of homogeneity of the material. Although the tubes were called "seamless," they showed a longitudinal seam upon being machined, an evidence of their non-uniformity. An attempt was made to anneal some

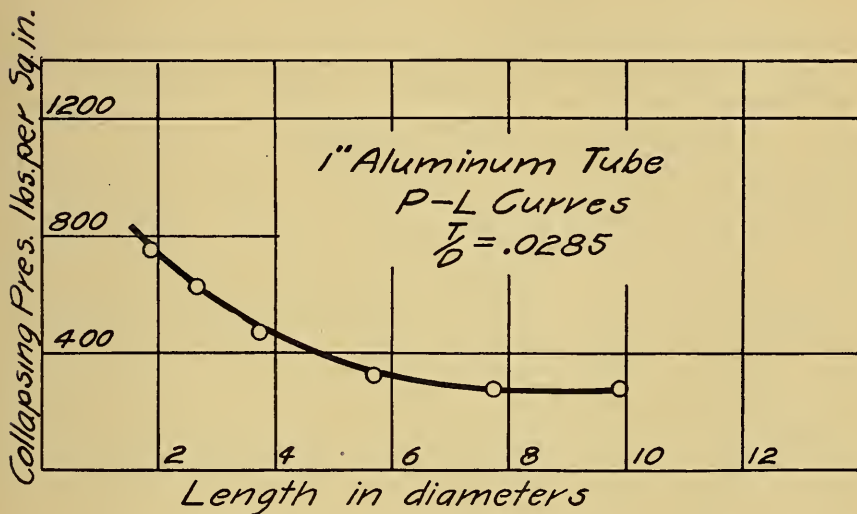


FIG. 12. PRESSURE-LENGTH CURVE FOR ONE-INCH ALUMINUM TUBE WITH THICKNESS OF $\frac{t}{d} = .0285$

of the steel tubes to secure greater uniformity, but this introduced great irregularities of form. The tubes were packed inside an iron box and heated to about 1400 degrees F. (measured by a thermocouple) in the large gas furnace of the Mechanical Engineering Department and then allowed to cool slowly. The circular tubes came out quite elliptical, and annealing was therefore abandoned. The tubes used were purchased through dealers in the open market. The steel tubes were Shelby cold drawn steel tubes made by the National Tube Company. As these tubes are not always straight and are liable to vary in wall thickness, the dealers kindly helped us in selecting tubes of as perfect form as could be found in stock. Through the kindness

of the laboratory of Applied Mechanics of this University we got the following elastic constants of the steel and brass:

For steel, Young's modulus	. .	29.94×10^6	pounds per sq. in.
Yield point	. . .	5.96×10^4	pounds per sq. in.
Ultimate strength	. .	7.25×10^4	pounds per sq. in.
For brass, Young's modulus	. .	13.90×10^6	pounds per sq. in.
Yield point	. . .	7.45×10^4	pounds per sq. in.
Ultimate strength	. .	9.90×10^4	pounds per sq. in.

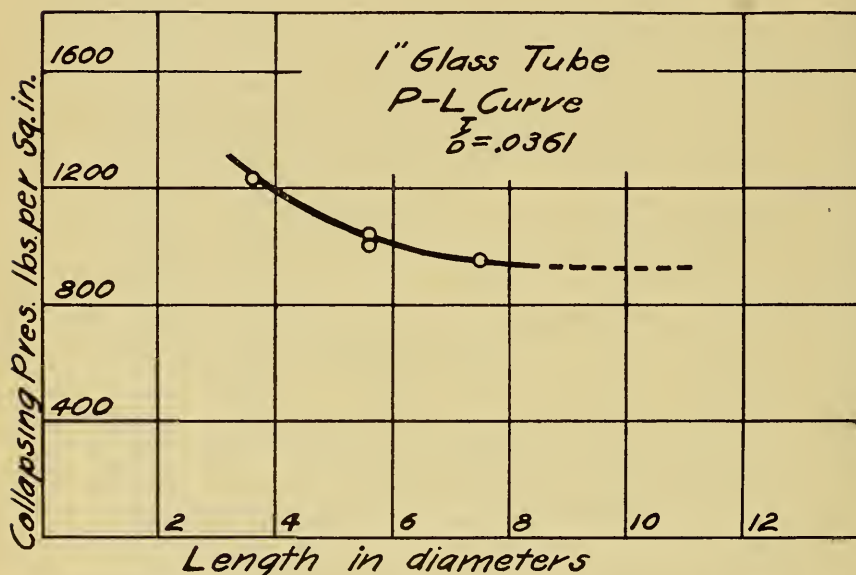


FIG. 13. PRESSURE-LENGTH CURVE FOR ONE-INCH GLASS TUBE WITH THICKNESS OF $\frac{t}{d} = .0361$

7. *Conclusions.*—A study of the curves leads to the following conclusions:

a. The pressure-length curves for different thicknesses and diameters are similar in shape. The series of curves for the one-inch and two-inch steel, the three-quarter-inch brass and the inch aluminum tubes show similar forms. The curves for glass and hard rubber differ from the others, the curve for the glass tubes bending more slowly, and that for the hard-rubber bending at a shorter length and

more rapidly. The number of experiments on these last two materials is, however, too small to justify much generalization.

b. By taking, in the case of the metal tubes, the point for the length of six diameters, and drawing a hyperbola through this point using the equation $p' = p \frac{L}{l}$, there is in most cases a satisfactory agreement between the observed and the calculated curves for lengths of tubes less than six diameters. In Figs. 15, 16, and 17, the parts of

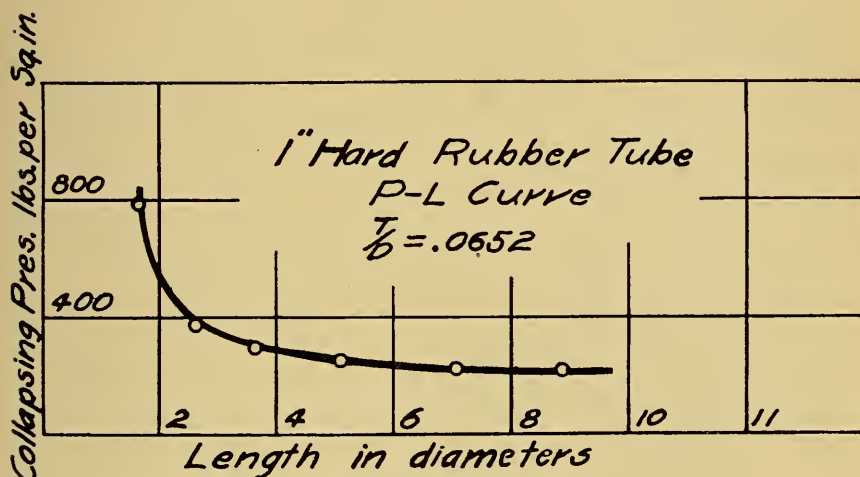


FIG. 14. PRESSURE-LENGTH CURVE FOR ONE-INCH HARD RUBBER TUBE WITH THICKNESS $\frac{t}{d} = .0652$

the curves for lengths less than six diameters have been reproduced from Figs. 8, 9, and 10, and the calculated hyperbolas have been drawn on them. In a few cases the agreement of the two curves is very close; in one case the two curves are identical on the scale of the figure. In Figs. 18, 19, and 20, the observed curve and the calculated curves have been drawn and continued to lengths greater than six diameters. It is seen that the hyperbolas thus calculated show pressures less than those given by experiment. This length of six diameters, then, appears to be a "critical length." The experimental curve at this length bends rapidly towards the horizontal, particularly in the case of the thicker tubes. In the case of the thinnest steel tubes, both for those of one-inch and of two-inch diameters, that is, for tubes

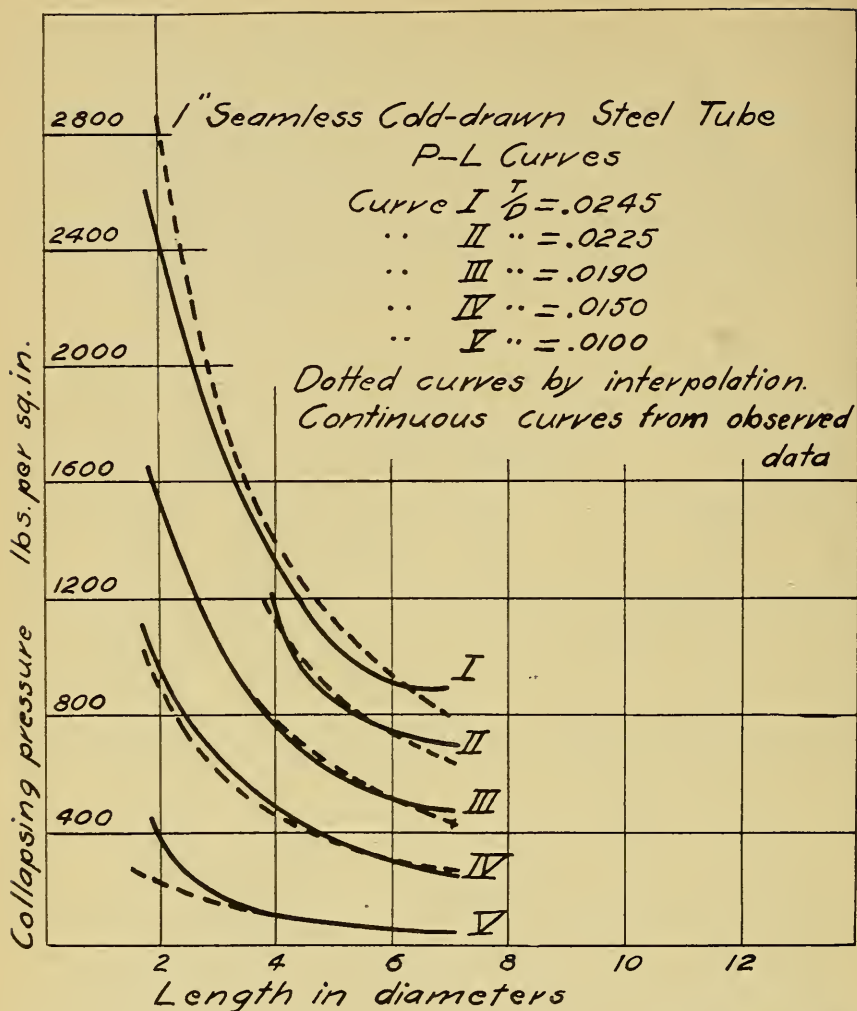


FIG. 15. PRESSURE-LENGTH CURVES OF ONE-INCH STEEL TUBES, PARTS OF THE CURVES FOR LENGTHS LESS THAN SIX DIAMETERS, COMPARED WITH HYPERBOLAS

in which the ratio of $\frac{t}{d}$ is about .001, the agreement with the hyperbola is not so exact; indeed, the maximum bend seems to occur at a much shorter length. While the curves for these very thin tubes show much uniformity, and the same characteristics are found in tubes of

2" Seamless Cold-drawn Steel Tube P-L Curves

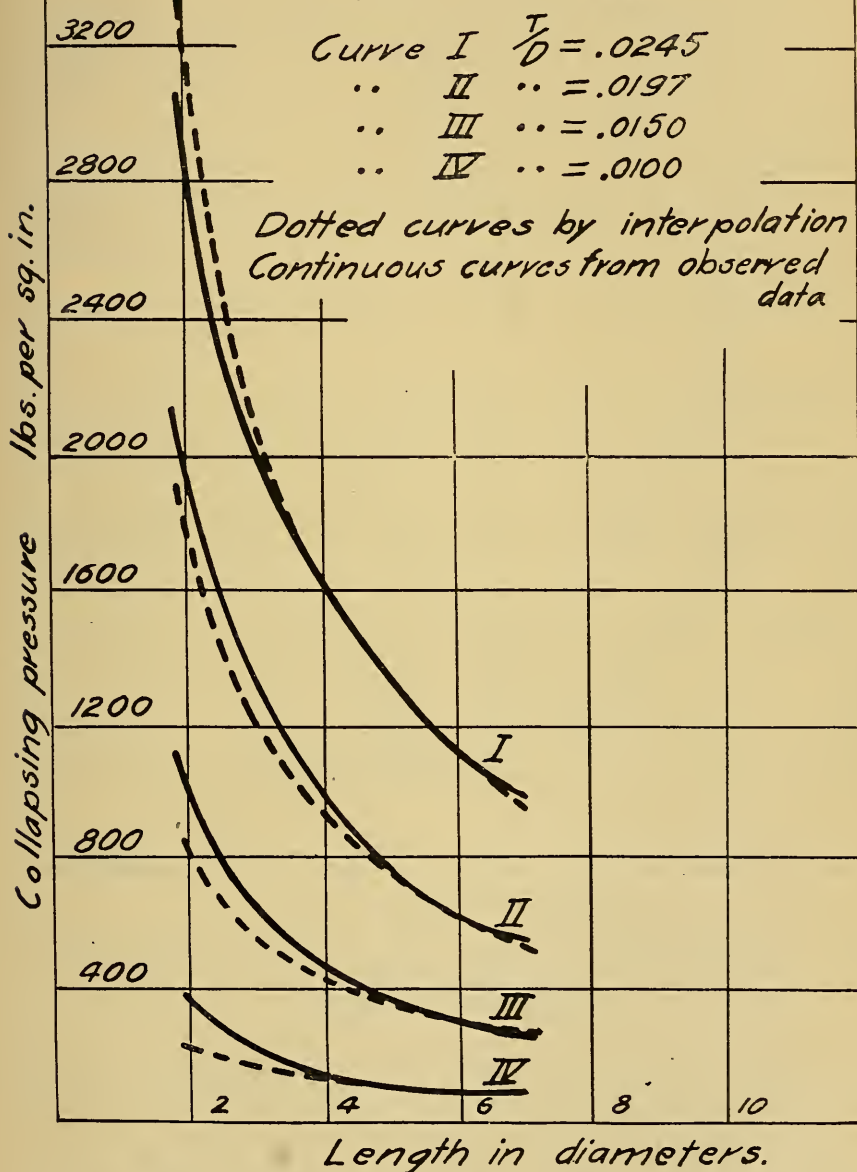


FIG. 16. PRESSURE-LENGTH CURVES OF TWO-INCH STEEL TUBES, PARTS OF THE CURVES FOR LENGTHS LESS THAN SIX DIAMETERS, COMPARED WITH HYPERBOLAS

different diameters, the percentage of error with such low pressures of collapse is necessarily greater.

c. The experimental curves of this investigation are not in agree-

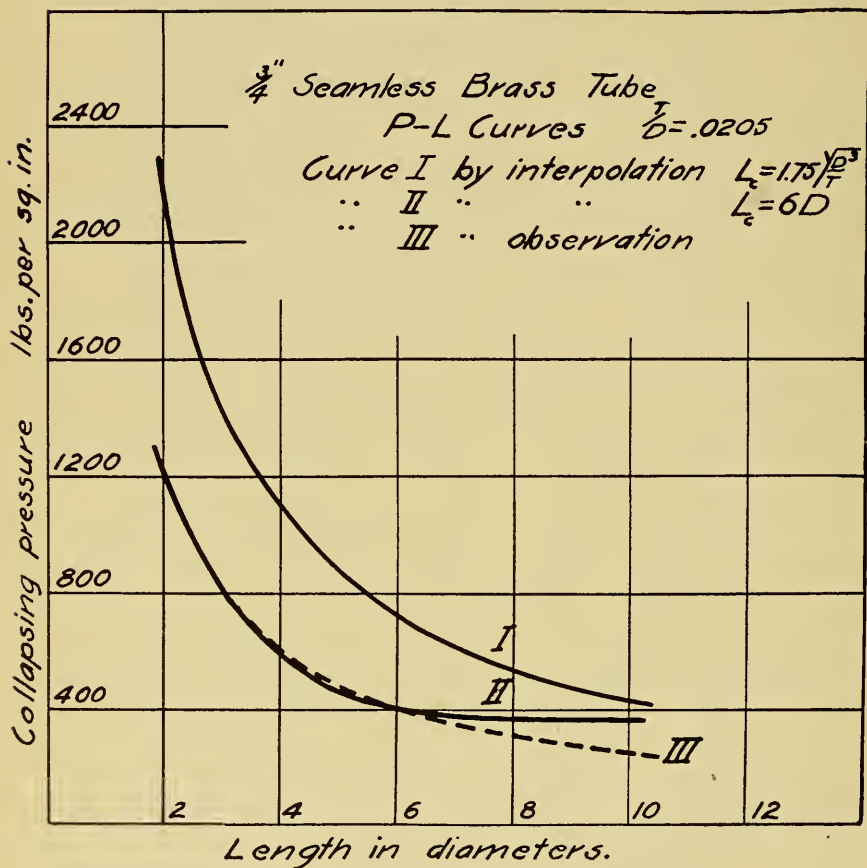


FIG. 17. PRESSURE-LENGTH CURVES OF THREE-QUARTER-INCH BRASS TUBES, PARTS OF THE CURVES FOR LENGTHS LESS THAN SIX DIAMETERS, COMPARED WITH HYPERBOLAS

ment with the Southwell formula $L = k \sqrt{\frac{d^3}{t}}$ which Cook has used.

This is shown in typical cases in Figs. 18, 19, and 20, where the curves for $L = 1.75 \sqrt{\frac{d^3}{t}}$ are drawn for the one-inch and three-inch steel tubes

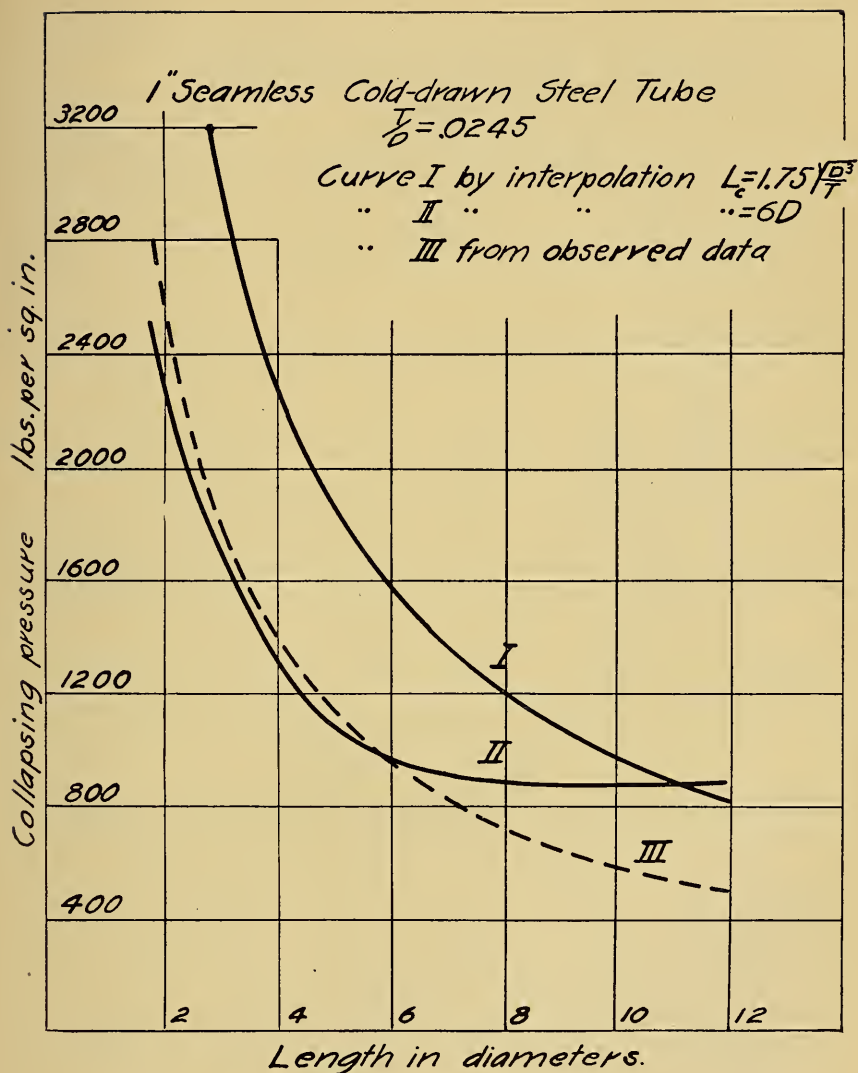


FIG. 18. PRESSURE-LENGTH CURVE OF ONE-INCH STEEL TUBE. EXPERIMENTAL CURVE COMPARED WITH TWO CALCULATED CURVES

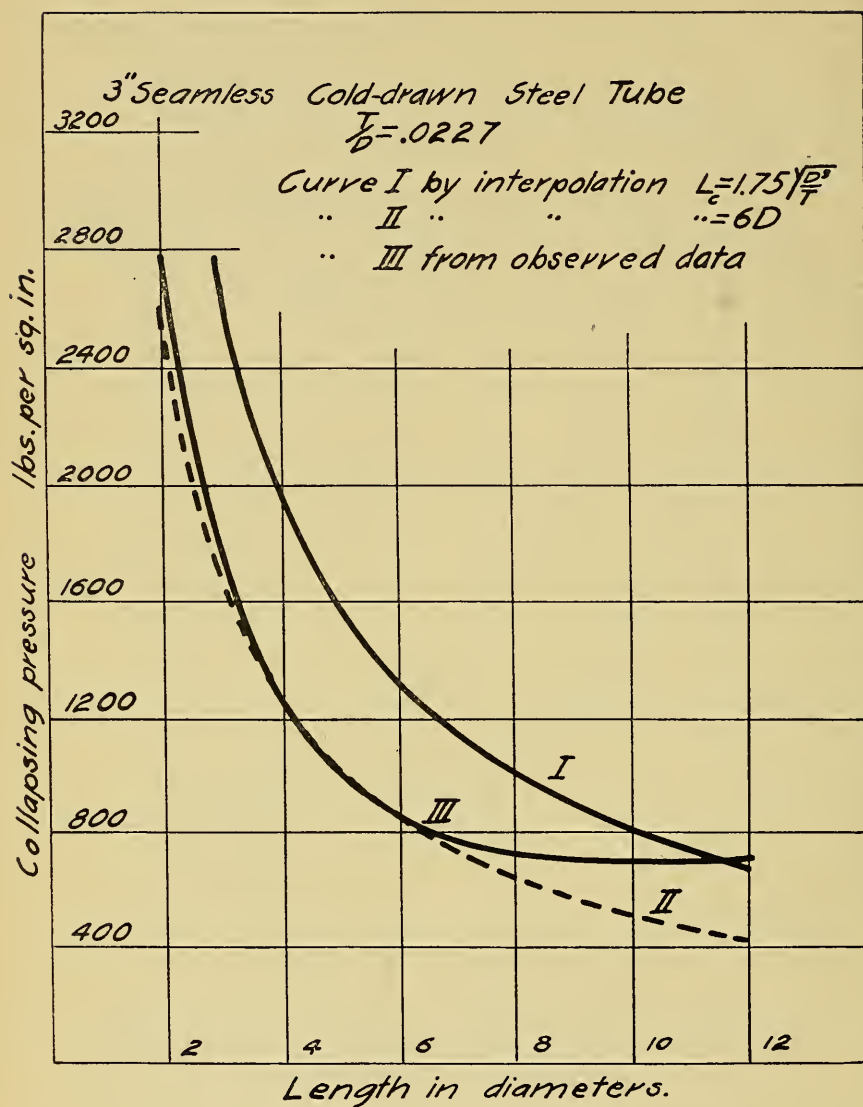


FIG. 19. PRESSURE-LENGTH CURVE OF THREE-INCH STEEL TUBE COMPARED WITH TWO CALCULATED CURVES

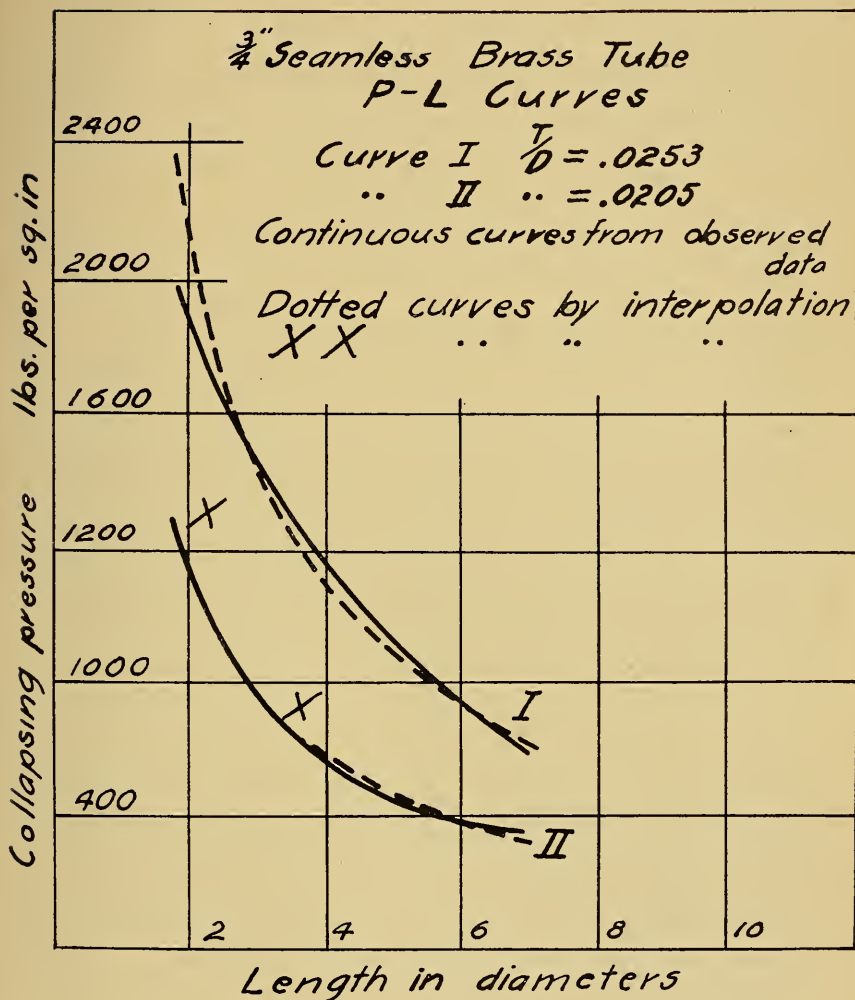


FIG. 20. PRESSURE-LENGTH CURVES OF THREE-QUARTER-INCH BRASS TUBES
 COMPARED WITH TWO CALCULATED CURVES

having a ratio $\frac{t}{d}$ of .0245 and .0227, and for the three-quarter-inch brass tube having a ratio $\frac{t}{d}$ of .0205. In the same figures the rectangular hyperbolas are drawn through the experimental point for a length of six diameters. If the formula $L = k \sqrt{\frac{d^3}{t}}$ is written in the form

$$L = \frac{kd}{\sqrt{\frac{t}{d}}} \quad . \quad . \quad . \quad . \quad . \quad . \quad (11)$$

the critical length should increase directly as the diameter and inversely as the square root of $\frac{t}{d}$. It is evident that the length corresponding to the critical bend does vary as the diameter of the tube, but the curves do not show that the thinner tubes have longer critical lengths. Indeed our curves for the very thin one-inch and two-inch steel tubes rather indicate a shorter critical length than six diameters. To obtain reliable results on very thin tubes, facilities are required for testing tubes of large diameters, so that small variations in thickness and material may not affect the final results.

The experimental curves for these metal tubes for which the ratio $\frac{t}{d}$ is between .025 and .015 show that there is a "critical" bend at the length of about six diameters, the part of the curve for shorter lengths being approximately a rectangular hyperbola. Beyond this critical bend the curve approaches more or less rapidly a straight line parallel to the axis of lengths. These results, in connection with previous results on the collapsing pressures of long tubes, make it possible to calculate from the data with reasonable approximation the collapsing pressures of short tubes of certain thicknesses.

TABLE 1
ONE-INCH STEEL TUBES
INSIDE DIAMETER .942 INCHES

Length in Inches	Length in Diams.	$\frac{t}{d}$	Pressure of Collapse Lb. per Sq. In.	Number of Lobes
1.72	1.8	.0245	2490	4
1.74	1.8	.0190	1600	3 or 4
1.785	1.86	.0150	1030	3 or 4
1.78	1.87	.01	420	4
2.63	2.73	.0245	2060	3
2.63	2.74	.0190	1180	3
2.63	2.74	.0150	670	3
2.63	2.75	.010	200	3
3.69	3.82	.0245	1390	2 or 3
3.69	3.84	.0190	800	3
3.69	3.85	.0150	550	3
3.69	3.9	.010	120	3
3.72	3.9	.0225	1170	3
5.63	5.85	.0245	920	2
5.63	5.73	.0225	730	2
5.63	5.85	.0190	520	2
5.63	5.9	.0150	320	2
5.63	5.9	.010	115	2
5.5	5.6	.0227	675	3
7.5	7.8	.0245	900	2
7.59	7.7	.0225	710	2
7.56	8.	.010	50	2
7.44	7.8	.015	210	2
7.62	8.	.015	190	2
7.62	8.	.019	420	2
9.6	10.	.0245	900	2
9.56	9.75	.0225	625	2
9.6	10.	.019	440	2
9.56	10.	.015	150	2
11.6	12.	.0245	750	2
11.63	11.85	.0225	600	2
11.63	12.	.019	375	2
11.6	12.1	.015	145	2

TABLE 2
TWO-INCH STEEL TUBES
INSIDE DIAMETER 1.873 INCHES

Length in Inches	Length in Diams.	$\frac{t}{d}$	Pressure of Collapse Lb. per Sq. In.	Number of Lobes
3.91	2.	.0245	2870	3
3.88	2.	.0197	1770	3
4.	2.1	.015	920	3
3.88	2.1	.010	350	2
5.94	3.1	.0245	2100	3
5.94	3.1	.0197	1250	3
5.94	3.1	.015	580	3
5.94	3.1	.010	150	2
7.8	4.1	.0245	1630	2
7.63	4.	.0197	980	2
7.63	4.	.015	540	2
7.66	4.	.010	135	2
11.34	5.9	.0245	1080	2
11.34	6.	.0197	590	2
11.34	6.	.015	260	2
11.34	6.	.010	70	2
15.13	7.9	.0245	960	2
15.13	7.9	.0197	510	2
15.13	8.	.0150	220	2
15.13	8.	.0099	55	2
18.94	9.9	.0245	794	2
23.0	12.5	.0245	780	2
27.8	14.5	.0245	855	2

TABLE 3
THREE-INCH STEEL TUBES
INSIDE DIAMETER 2.87 INCHES

Length in Inches	Length in Diams.	$\frac{t}{d}$	Pressure of Collapse Lb. per Sq. In.	Number of Lobes
5.88	2.	.0227	2640	4
9.12	3.1	.0227	1960	..
12.14	4.1	.0227	1390	2
12.17	4.1	.0227	1040	2
12.19	4.1	.0227	1010	2
15.06	5.1	.0227	873	2
15.06	5.1	.0227	910	2
18.13	6.2	.0227	756	2
18.28	6.2	.0227	895	2
24.13	8.2	.0227	684	2
24.25	8.2	.0227	770	2

TABLE 4
THREE-QUARTER-INCH BRASS TUBES
INSIDE DIAMETER .751 INCHES

Length in Inches	Length in Diams.	$\frac{t}{d}$	Pressure of Collapse Lb. per Sq. In.	Number of Lobes
1.56	2.	.0138	310	3
1.56	2.	.0138	400	3
1.56	2.	.0205	1150	3
1.56	2.	.0253	1900	3
2.28	3.	.0138	345	..
2.28	3.	.0138	300	..
2.28	2.9	.0205	920	..
2.25	2.9	.0253	1470	..
2.25	2.9	.0253	1310	..
3.	3.9	.0253	1400	..
3.	3.9	.0253	1360	..
3.	3.9	.0276	1420	..
4.	5.7	.0205	409	2
4.	5.7	.0205	429	2
4.56	5.9	.0253	810	2
4.63	6.	.0253	700	2
5.69	7.4	.0205	348	2
6.	7.8	.0205	550	2
7.5	9.8	.0205	362	2
7.5	9.8	.0205	388	2

TABLE 5
ONE-INCH ALUMINUM TUBES
INSIDE DIAMETER .944 INCHES

Length in Inches	Length in Diams.	$\frac{t}{d}$	Pressure of Collapse Lb. per Sq. In.	Number of Lobes
1.18	1.9	.028	750	4
2.59	2.7	.0286	630	3
3.63	3.7	.0285	470	3
3.63	3.7	.029	475	3
5.53	5.7	.0288	330	2
7.53	7.8	.029	280	2
9.59	9.9	.0295	280	1

TABLE 6
ONE-INCH GLASS TUBES
INTERNAL DIAMETER .96 INCHES

Length in Inches	Length in Diams.	$\frac{t}{d}$	Pressure of Collapse Lb. per Sq. In.	Condition of Collapsed Tube
3.59	3.7	.036	1230	practically pulverized
5.63	5.8	.036	1040	" "
5.03	5.8	.036	1000	" "
7.47	7.7	.036	950	" "

TABLE 7
ONE-INCH HARD RUBBER TUBES

Length in Inches	Length in Diams.	$\frac{t}{d}$	Pressure of Collapse Lb. per Sq. In.	Condition of Collapsed Tube
1.69	1.6	.0643	790	One side caved in and broken in very fine parts.
2.63	2.5	.0654	370	In irregular parts
3.63	3.4	.0650	290	"
5.56	5.2	.0654	250	"
7.56	7.1	.0652	220	"
9.55	8.9	.0645	220	"

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UNIVERSITY OF ILLINOIS BULLETIN

ISSUED WEEKLY

Vol. XIV

JUNE 18, 1917

No. 42

Entered as second-class matter Dec. 11, 1912, at the Post Office at Urbana, Ill., under the Act of Aug. 24, 1912.]

PERCENTAGE OF EXTRACTION OF BITUMINOUS COAL WITH SPECIAL REFERENCE TO ILLINOIS CONDITIONS

BY

C. M. YOUNG

ILLINOIS COAL MINING INVESTIGATIONS

COÖPERATIVE AGREEMENT

(This Report was prepared under a Coöperative Agreement between the Engineering Experiment Station of the University of Illinois, the Illinois State Geological Survey, and the U. S. Bureau of Mines)



BULLETIN No. 100

ENGINEERING EXPERIMENT STATION

PUBLISHED BY THE UNIVERSITY OF ILLINOIS, URBANA

EUROPEAN AGENT
CHAPMAN & HALL, LTD., LONDON

THE Engineering Experiment Station was established by act of the Board of Trustees, December 8, 1903. It is the purpose of the Station to carry on investigations along various lines of engineering and to study problems of importance to professional engineers and to the manufacturing, railway, mining, constructional, and industrial interests of the State.

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The present bulletin is issued under a coöperative agreement between the Engineering Experiment Station of the University of Illinois, the State Geological Survey, and the United States Bureau of Mines. The reports of this coöperative investigation are issued in the form of bulletins by the Engineering Experiment Station, the State Geological Survey, and the United States Bureau of Mines. For bulletins issued by the Engineering Experiment Station, address Engineering Experiment Station, Urbana, Illinois; for those issued by the State Geological Survey, address State Geological Survey, Urbana, Illinois; and for those issued by the United States Bureau of Mines, address the Director, United States Bureau of Mines, Washington, D. C.

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BY
C. M. YOUNG
ASSISTANT PROFESSOR OF MINING RESEARCH

ENGINEERING EXPERIMENT STATION

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PERCENTAGE OF EXTRACTION OF BITUMINOUS COAL WITH SPECIAL REFERENCE TO ILLINOIS CONDITIONS

INTRODUCTION

1. *Preliminary Statement.*—The purpose of the discussion presented in this bulletin is to record the results now being obtained in recovering coal in the mines in Illinois and in other bituminous coal mining districts of the United States. A brief discussion is also presented with reference to recovery in the principal European countries. Where the methods employed are now producing an unusually good percentage of extraction, the conditions under which the mining is carried on are described in considerable detail with the belief that they may suggest changes in practice which will be helpful to those who are now endeavoring to recover a greater percentage of the coal in the ground.

Most of the data presented were obtained from those operating the mines, and represent, therefore, calculations or estimates based upon thorough familiarity with conditions. Some of the methods by which high extraction is attained in other districts are described with the hope that the coal producers of Illinois may find herein suggestions which will prove helpful in their efforts to attain higher recovery.

It has been impossible to include in a single publication all the material available concerning the physical conditions encountered and the methods adopted in the various coal fields, but there will be found in the bibliography a list of books and articles in which these subjects are covered in greater detail. It is the present purpose to begin at an early date a more extended investigation of the plans and dimensions of mine workings in Illinois with reference to the cost of production and the percentage of extraction.

2. *Acknowledgments.*—The writer wishes to acknowledge his indebtedness to PROFESSOR H. H. STOEK, head of the Department of Mining Engineering, University of Illinois, and to MR. F. W. DEWOLF, Director of the Illinois State Geological Survey, and also to MR. G. S. RICE, Chief Mining Engineer, United States Bureau of Mines, under whose direction the work of the Illinois Coal Mining

Investigations is being carried on. Professor Stoek has been especially helpful in the collection of material for the present bulletin. Many operators and engineers throughout the country have contributed statements concerning the districts with which they are familiar. The state mine inspectors have assisted in the work by suggesting mines at which particularly good records of extraction have been made and also mines at which new methods are being tried with a view of increasing the percentage.

3. *Summary.*—The facts and information presented in this bulletin include:

(1) A general statement of the importance of the problem of increasing the percentage of extraction of the coal in the ground in order to utilize the coal resources to a greater extent than at present, and, if possible, to decrease the cost of producing coal; also an account of previous efforts made to compile data upon this subject.

(2) A statement with reference to the conditions which have influenced the development of American coal mining methods and which must be considered in changing these methods in order to obtain more nearly complete recovery.

(3) A record of the recovery of coal in Illinois in the past, and a discussion of the efforts now being made to increase the percentage of extraction.

(4) An account of methods adopted in other states and in certain European countries by which higher percentages of extraction are being obtained.

(5) A brief history of the development of English mining practice, upon which American practice is founded.

(6) A short bibliography with reference to the subject of coal mining methods.

4. *Conclusions.*—A summary of conclusions suggested by a study of the data and information contained herein is presented as follows:

(1) In general in America probably one ton of coal has been left in the mine for every ton brought to the surface.

(2) An effort is being made in many sections of the United States and in a number of Illinois mines to decrease this loss of coal.

(3) The low percentage of recovery in the United States is largely due to economic conditions and to efforts to produce cheap coal.

(4) Where economic conditions have been favorable, percentages of recovery have been obtained in the United States quite as high as in any of the foreign countries in which usually the economic conditions have not been such as to make the production of cheap coal the determining element in the choice of a method.

(5) The low price at which much of the coal land in the United States has been bought has not offered an inducement to save the coal.

(6) The best results in recovery are now being obtained in districts where the value of coal land is high.

(7) As a general rule better extraction is being obtained in West Virginia and Pennsylvania than in the Middle West.

(8) In view of the results being obtained in some other districts, under conditions no more favorable than those in Illinois, the percentage of extraction in Illinois should be increased.

(9) The best results are being obtained by the larger and stronger companies which can afford to plan for the future.

(10) The low value of the smaller sizes of coal in the past has been a drawback to pillar drawing, because very often pillar coal has contained more of the small sizes than room coal. With the increasing use of small sizes in mechanical stokers, the price will undoubtedly advance to nearly the same level as that of the larger sizes; thus this drawback to greater recovery will gradually disappear.

(11) One of the reasons why newer methods have not been generally tried is to be found in the prejudice, too common in coal mining practice, against innovations, and in the fact that mining methods have been based largely upon previous practice in other countries or in other states.

(12) Many of the attempts to draw pillars have been unsystematic. Upon such unsystematic work are based many of the opinions concerning the technical and commercial practicability of pillar drawing and the prejudices against it.

(13) Subsidence of the surface must be regarded as a necessary accompaniment of mining. Instead of trying to prevent

subsidence, the pillars should be removed systematically so that the surface subsidence will occur uniformly and not in isolated spots. Although there may be a temporary disturbance of the surface, after a short time its condition will be as good or nearly as good as before the mining.

(14) In Illinois, at the present time, more than fifty per cent of the coal is frequently left in the ground in an effort to prevent squeezes and subsidence; even then it is not at all certain that the desired result is accomplished.

(15) The best results may be obtained by driving room entries to their full length, then by beginning the rooms at the inby end of the entry, in order that pillar drawing may begin as soon as the inby room is finished.

(16) To be effective, pillar drawing must begin as promptly as possible after the rooms are worked out.

(17) Where pillars are left to be drawn subsequently, the coal is usually lost, because the pillars are crushed through squeezes, or because it is not found economical or convenient to take the coal out and at the same time to keep up the output of the mine with the comparatively small amount of coal left. In other words, unless pillar drawing follows very closely after the first working, very little pillar coal is obtainable.

(18) In many districts poor top has prevented taking out the full thickness of the coal, and one of the great losses is that due to coal left in the roof. This loss has been overcome in some cases very successfully, and should be carefully studied.

(19) The reported percentages of extraction are usually too high because, in estimating, often only the section mined is considered and no account is taken of top or bottom coal left unmined. Also frequently only limited areas of the mine are considered instead of the mine as a whole.

(20) At different mines in the same region where physical conditions are practically the same, the mining methods vary widely with regard to length of rooms, number of rooms in a panel, thickness of barrier pillars, etc. This variation in practice suggests the advisability of a detailed study to determine, if possible, a standard method for a given set of conditions.

CHAPTER I

MINING METHODS AND CONDITIONS IN RELATION TO EXTRACTION

5. *Introduction.*—The subject of the percentage of coal extracted from the mines in the United States has received very meager attention, except in the case of individual mines or companies. The only comprehensive official study of an extended coal mining area has been in the anthracite district of Pennsylvania, where the high value of the coal and the knowledge that the supply is limited early stimulated an interest in the subject. This interest led to the appointment of the Coal Waste Commission which reported in 1893.*

In 1905 H. H. Stoek† published a table of coal pillar data which contained percentages of extraction gathered largely by correspondence. See Table 1, page 23.

In 1914 A. W. Hesse‡ collected as much information as possible on this subject, which is summarized in Table 2, page 24.

In previous bulletins of the Coöperative Coal Mining Investigations tables of pillar data and percentages of extraction were given. These are summarized in Table 3, pages 25, 26, 27.

Doubtless many of the figures in these tables and others on the percentage of recovery are open to question, but they represented the best and most nearly complete information available when they were published. There are several reasons for questioning the accuracy of the figures on extraction. Chief among them is the fact that estimates are usually based upon areas which are too small or upon insufficient data. A single panel or a single lease is sometimes used as a unit upon which to base estimates, and often the areas thus selected are favorably located. While the estimates may represent results obtained with the given method of mining, they by no means represent the average results for the mine as a whole. The panel selected for measurement is usually one in which there has been no squeeze, while all about it there may be squeezed areas in which large amounts of coal have been lost. Many estimates are

*Report of Commission Appointed to Investigate the Waste of Coal Mining with the View to the Utilizing of the Waste, 1893.

†Mines and Minerals, Vol. 26, p. 107, 1905, and International Library of Technology, Vol. 150, par. 40, p. 60.

‡"Maximum Coal Recovery," W. Va. Min. Inst., June 3, 1914; Coll. Eng., Vol. 35, p. 13; and Coal Age, Vol. 5, p. 1051.

based upon possible future recovery from pillars, which may or may not be obtained.

Estimates covering extraction frequently do not take account of top and bottom coal left in the mine, and the values reported often refer only to the section of the coal actually mined. In mining a coal bed ten feet thick, for instance, two feet of top coal may be left unmined. The maximum percentage of extraction from mining eight feet, in this case, would be eighty per cent of the total coal in the bed. If, then, fifty per cent of the eight feet mined is obtained, only forty per cent of the total coal in the bed is recovered.

The only accurate method of estimation is to divide the actual amount of coal mined, as determined by the tonnage for which the miner is paid, by the amount of coal in the ground as determined by multiplying a given area by the average thickness shown in a large number of sections of the bed.

Even where great care is exercised, results are often subject to errors. The causes for these have been outlined by Smyth* as follows: Inaccuracies in railroad weights of possibly five to ten per cent, inaccuracies in estimation of coal used at the mines frequently amounting to ten per cent, inaccuracies in estimating the mean thickness of the bed amounting, even in very uniform beds, probably to five per cent, difficulties of obtaining final figures until a mine is worked out.

The present condition of the coal mining industry in this country is a natural result of the course and character of its development. In general, only those beds and even parts of beds have been worked, the exploitation of which would result in the largest immediate profits. Those methods of mining which were cheapest and which promised the largest profit on the coal produced have been followed, often without regard to the possible injury of the mine or the resulting loss of coal. There has been, moreover, no restriction of market, and in many cases districts have been opened when there has been very little demand for coal in the surrounding territory, but when conditions of operation and transportation have been such as to make it possible for coal from these districts to enter markets already supplied. The result has been cheap coal, produced by wasteful methods.

Another result has been over-development of the industry. The opening in nearly all districts of too many mines has resulted in the

* Smyth, John G., Personal Communication.

idleness of many mines during a large part of each year with the accompanying increase in the cost of production. In dull periods coal has frequently been sold for less than the cost of production in order that mines might be kept in operation and certain fixed charges met. This subject was taken up by Bush and Moorshead in 1911 in a paper* before the American Mining Congress in which it was said that the production in this country exceeded the consumption first in 1891, and that the difference between consumption and capacity for production had steadily increased. The strike of 1910 in Illinois, Indiana, and the Southwest emphasized the over-capacity of the mines of that region. Though the mines of Illinois were idle during six months of the year, the production of 45,900,246 tons was only ten per cent less than the production of the previous year. The mines of Oklahoma, Arkansas, and Missouri were also idle during six months of 1910 because of the strike, but the production showed an average decrease of only twenty per cent. It was also said that the possible capacity of West Virginia mines was seventy-five per cent more than the total production, that the output in the Pittsburgh and the No. 8 Ohio districts was reduced to thirty per cent of the normal production during the three or four months of each year when navigation on the lakes was closed, and that few properties during the three preceding years—1909, 1910, and 1911—had been operated more than 225 working days per year.

This over-production, with its small profit or even loss in the operation of mines, results in a natural tendency to employ only those methods which will insure cheap coal. It is natural, also, that under these conditions there should exist an attitude of hesitancy with regard to the adoption of new or different methods. Neither the coal producer nor the public has as yet become aroused to the full realization of the fact that the natural resources of the country are not inexhaustible. The coal mining engineer of America accordingly, has not had as his problem the development of methods of extraction which would result in the largest percentage of ultimate recovery, but rather the development of methods which would result in the lowest cost of production. In many cases, however, as is shown by the detailed descriptions given later, where economic conditions have seemed to warrant it, methods have been developed by

* Bush, B. F., and Moorshead, A. J., "The Condition of the Bituminous Coal Industry," Proc. Amer. Min. Cong., p. 246, 1911.

American engineers and coal producers which have given a percentage of recovery equal to that secured in any European country.

The fact should be borne in mind, when comparisons are made between mining methods in different countries, that, while it is true that the percentage of extraction is less in this country than in most of the European countries, the cost of coal to the consumer and the profit to the producer are also less.

The subject of the comparative cost of production of coal and of the comparative profits realized in Great Britain and in the United States was taken up by Rice* substantially as follows: the average value of coal in the United States on cars at the mine in 1913 is reported as \$1.18 per short ton for bituminous coal and \$2.13 per short ton for anthracite. In Wales, in 1913, the average value per short ton at the mines for all kinds of coal was \$2.55, and in Great Britain as a whole, \$2.21. In the German Empire the average value for all kinds was \$2.27, and for Westphalia it was \$2.37 per short ton. Net mining profits in Great Britain and in Germany are between twenty-five and fifty cents per ton, while profits in the United States for bituminous coal are probably not more than five cents per ton.

It is a matter of course that more expensive methods of mining cannot be adopted without increasing the cost of the coal, and under the conditions which have prevailed in the coal industry for many years there could be no material increase in the cost of coal to the producer without a corresponding increase in the selling price. The prevailing opinion, however, that the percentage of recovery cannot be greatly increased without an increase in the cost of production is questionable, and certainly this increase in cost would not be as great as is generally believed. This is a matter which can be conclusively determined only by actual trial of new methods extending over a sufficient period to insure the reliability of the results. The fact that the adoption of methods which result in an increase in the percentage of extraction has been possible in some districts with little or no increase in cost at least furnishes a reason for thinking that similar changes could be made in other districts with similar results.

Careful planning of operations over long periods and steady working are necessary in order to obtain a high percentage of ex-

*Rice, G. S., "Mining Costs and Selling Prices of Coal in the United States and Europe, with Special Reference to Export Trade," Second Pan-American Scientific Congress.

traction. At present these conditions are impossible in many districts and can be attained only by centralized control of production and selling price, which will provide against alternation of idle and rush periods with the disorganization which accompanies them. Under existing conditions it is feared by operators that the necessary co-operation would be interpreted and attacked as a violation of anti-trust laws. In some of the European countries syndicates working in coöperation with the governments regulate the output of the mines and the selling price of coal with results which are said to be highly satisfactory and conducive to a high recovery.*

One of the chief commercial factors affecting the choice of a method has been the cost of coal in the ground. This has generally been very low, and the loss of coal, therefore, has not been considered a serious matter. Even at the present time the value of coal rights in the southern Illinois field, where the No. 6 bed is worked, is estimated at not more than \$100 to \$150 an acre, and it has been only a very short time since such coal rights could be purchased for less than \$50 an acre. The thickness of this coal is somewhat variable, being in some places fourteen feet or more, but, if we assume that only about seven feet is worked, the output will amount to about 12,000 tons per acre and the cost of coal in the ground will be about one cent per ton. A great deal of the coal in the state, however, has been bought at a very much lower figure. In some cases also there is a second bed of coal which will be available later, and when this is considered, the cost of coal in the ground will be much less than one cent per ton.

This phase of the subject was discussed by Rice,† in 1909, as follows:

"The influencing conditions causing the great losses that are at present incurred are:

1. Cheapness of 'coal in place'; that is, in the seam.
2. Low market prices, resulting from extreme competition.
3. Character of the seam, roof, and floor as determining the method of mining.
4. Surface subsidence due to mining.
5. Interlaced boundary ownerships.

*Scholz, Carl, "The Economics of the Coal Industry," Proc. Amer. Min. Cong., p. 241, 1911.

†Rice, George S., "Mining-Wastes and Mining Costs in Illinois," Trans. Amer. Inst. Min. Eng., Vol. 40, p. 31, 1909.

6. Carelessness in mining operations.

The first two factors, taken together, are the controlling ones in most mining operations in influencing the choice of a mining system. The majority of Illinois operators are sufficiently progressive to find ways and means to take out practically all the coal under a given area if it could be made evident that it paid to do so. That many do not do all that can be done in this direction is apparent; but if, without unusual investment, a profit of operation could be shown in taking out all the coal over the profit made by present methods, the industry could undoubtedly find men to accomplish the task. In other words, from an engineering standpoint practically all the coal under a given area can be taken out. It is a question of cost.

“Cheapness of Coal in Place.—This is chiefly due to the great abundance of coal. Except in the barren northern one-fourth of the State, lying north of the outcrop of the coal-basin, the development of a tract depends primarily not on the possibility of finding coal in that particular locality, but on the question whether it is a suitable place, from a market standpoint, to open a mine, the thickness of seam and the quality of the coal being considered.

“The price of coal rights varies from \$10 per superficial acre in the middle part of Illinois, away from the mining centers to \$100 per acre near developed mines. Or, in the case of leasing, from 2 cents per ton run-of-mine hoisted, in the southern part of the State, to 5 cents in the northern part. The cost of the fee is relatively so much cheaper per ton than leasing that the latter system is not much used. The ownership of the coal by the operator is conducive to better mining, but relative to other items that go to make up the total, the cost of the ‘coal in place’ is so low as to be almost negligible. In central Illinois, in some cases, at a cost of only \$10 per acre, two workable seams, from 6 to 8 ft. thick, are obtained. Allowing only 50 per cent yield of the two seams, 13,000 tons would be produced per acre, the purchase cost thus being $\frac{1}{13}$ of a cent per ton, or about $\frac{1}{1000}$ of the total cost of production in central Illinois. In the Wilmington long-wall field the average cost of the coal rights is about \$50 per acre. The seam there, although it averages a trifle less than 3 ft. in thickness, produces about 5,000 tons per acre. The cost is therefore about 1 cent per ton in place, which is $\frac{1}{130}$ of the total cost of production. Hence, it may be seen that there is little incentive, from the standpoint of the purchase price of the coal, to save the latter in mining operations.”

The cost of coal rights has very greatly increased since Rice's discussion, and there is every reason to believe that the value of coal in the ground will be much greater in the future than it has been in the past. During most of the productive period, however, the coal in the ground over a considerable part of the State has been worth not more than one-tenth of a cent per ton, and under these circumstances the loss of coal has, naturally, not been considered a serious matter. What has been important, and still is, is the extraction of coal at low cost, and the subject of high recovery is one of increasing importance at the present time.

Every ton of coal left in the ground represents the loss of a possible profit. Every ton of such coal represents a loss in increased value. An acre of coal left in the ground at any time means the extraction of another acre at some later time when the value of coal in the ground will be greater. In other words, producers are now extracting coal, worth possibly \$150 an acre, which might be left until it would reach even a greater value if it were not for the fact that coal was wasted in the ground when it was worth only fifty dollars an acre.

A low percentage of extraction increases the cost of production, because, for a given output, the workings must cover a larger area. This involves longer haulage roads, and, consequently, a greater investment in rails and trolley wire, greater maintenance expense, greater consumption of power, lower output per unit of equipment, and lower output per man. With long haulage roads there is greater chance for derangements of the track or for falls of roof, which may cause the stopping of haulage until the trouble is removed or may result in wrecks if the trouble is not discovered in time. Another source of danger lies in the greater haulage speed which must be employed on long roads if the output is to be maintained.

The cost of ventilation is also higher, because these larger workings require a larger quantity of air to maintain safe conditions, and more power is required to circulate air through the longer passages. There is, moreover, a greater loss of power because of the more numerous stoppings, which are often inefficient.

Another difficulty accompanying the spreading of the workings over a large area is that of providing the intensive supervision which is highly desirable in coal mining, particularly where skilled workmen have been replaced by comparatively unskilled laborers, and

the pick has been replaced by explosives and mining machines. Unless the cost of operation is to be increased by the employment of a larger number of foremen or face bosses, this intensive supervision can be obtained only by concentration of the workings.

6. *Subsidence*.—One reason for the use of methods involving low extraction is the desire to maintain the original surface of the ground. Rice* discussed surface subsidence as follows:

“ The influence of this factor upon the yield results from the high value of Illinois lands for agricultural purposes. . . . If the long-wall system were applied to the thick seams, when applicable at all, it would cause a considerable derangement of the surface, and when the latter is so nearly level as the prairie-land of Central Illinois, it makes the question of subsidence a serious one. . . . However, until the agricultural land in the United States becomes insufficient to fill the needs of the population, which would be reflected in a continual increase of price for farming land, the money-loss from temporarily destroying the surface in places is relatively small, as compared with the selling price of the coal mined from the seam. Taking the average value of the surface at \$125 per acre, if 80 per cent be rendered worthless the immediate money-loss would be \$100 per acre. A seam 6 ft. thick would contain per acre 11,000 tons of coal in place, yielding, at 90 per cent, 9,900 tons. The damage done by practically destroying the surface would be only 1 cent per ton. If the land-prices should rise to an amount two or three times as great as the value stated, this loss would still not prohibit mining.”

As far as the long-wall district is concerned, very little if any damage has resulted from subsidence, and little attention has been given to the subject. The most noticeable effects are generally temporary, and farm operations are not hindered.

The subject of the relation of surface values to subsidence in Illinois has been considered by L. E. Young in Bulletin 17 of this series.† Fig. 1 is a map reproduced from this bulletin, on which the approximate values of farm lands are shown. Table 4, page 28, shows the relation between coal values and surface values. The land values indicated on the map and set forth in the table suggest that it might be possible, at least in many cases, to mine coal at a small profit even if the value of the surface were totally destroyed. There is no need, however, for assuming the permanent destruction of the surface or even its serious permanent injury. Generally any damage resulting from subsidence could be largely or wholly re-

* Rice, Op. cit., p. 40.

† Young, Lewis E., "Surface Subsidence in Illinois," Ill. Coal Min. Invest., Bulletin 17, p. 52, 1916.

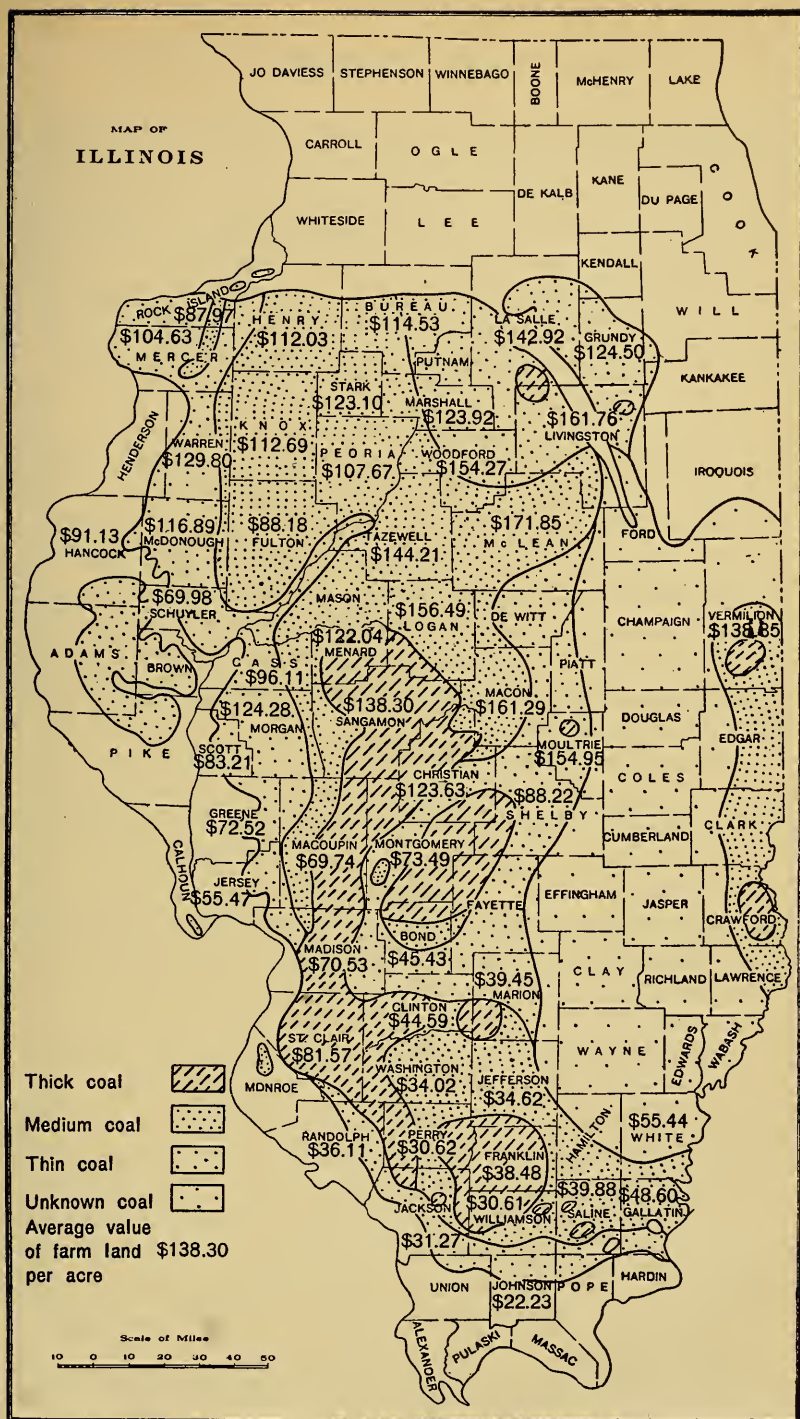


FIG. 1. MAP SHOWING THICKNESSES OF COAL AND VALUES OF FARM LANDS, AS GIVEN BY THE 1910 CENSUS REPORTS

paired, especially if it were accepted that mining is certain to result in subsidence and operations were so planned as to reduce the surface damage and the cost of restoration to the lowest possible amount.

When the coal producer owns nothing but the coal rights, unreasonable damages for surface subsidence are sometimes imposed. The measure of the damages is not always merely the decreased productive value of the land, nor the cost of restoring it to its former condition by artificial drainage; the formation of a small pond has often been claimed to lower the market value of a farm to a considerable extent, simply because it made the farm less sightly. Under these circumstances, operators naturally desire to avoid disturbing the surface for they know that an attempt will be made to recover damages and that, even if they escape the payment of exorbitant amounts, they will incur considerable expense in defending the suits. An effort, accordingly, is often made to conduct the mining operations in a manner which will not result in surface subsidence. The result is that the loss in the ground represents an important percentage of the coal, in many cases more than half that contained in the area worked.

It is very important that the allowable damages for surface subsidence be regulated by some law. This law should fix the damage payable by the coal producer, in case he is legally responsible, upon the basis of the actual damage done to the surface. Under such a law the operators would know the extent and character of their responsibility and could, without fear of excessive or unreasonable damages, proceed according to methods which would yield the highest possible percentage of extraction justifiable under such conditions.

7. *Squeezes*.—Closely related to, but not identical with, the subject of surface subsidence is that of squeezes. There may be subsidence without a squeeze, but with the conditions in Illinois a squeeze is usually followed by subsidence. The removal of a portion of a deposit throws additional weight upon the pillars left and if these pillars are not strong enough to support this additional weight, they will crack and crush, causing a movement of the overlying material. This movement is called a "squeeze," or sometimes a "creep." Large quantities of coal are often left in the mine in the form of pillars in an effort to prevent a squeeze, which may not only interfere with the operation of the mine and entail a loss of

the coal in the squeezed area, but large areas of the mine may become inaccessible for future economical working. A more nearly complete extraction of the coal properly carried out should, however, result in less damage from squeezes.

There are two ways in which a squeeze may be prevented or stopped; first, by the use of a support strong enough to prevent any movement of the overlying rock, and secondly, by a fracture of the rock above the excavated portion so that the weight on the pillars will not be sufficient to crush them.

The first method may be employed by either leaving natural supports (coal pillars) of sufficient size and strength to hold the roof without any movement, or by the use of artificial supports, such as timber or iron columns, or sand or culm filling. The cost of timber or iron would prohibit their use if a large percentage of coal was to be extracted. Filling is not generally non-compressible, but it occupies most of the space from which coal has been removed, prevents any scaling off of pillars, and eliminates any possibility of movement of pillars. The filling method, however, is hardly to be considered feasible in Illinois because of the cost. In the Upper Silesian coal field where this method is most extensively used, the cost is from twelve to eighteen cents per ton.*

Another difficulty arises from the fact that the material would have to be flushed into the mine with water, and then the water would have to be pumped out. This water would probably have an injurious effect on the clay bottom. The material, moreover, would have to be brought from a distance unless the value of the land should be so small as to permit the use of material from the neighboring surface, and this condition would rarely prevail in Illinois.

In the leaving of coal pillars of sufficient strength to prevent roof movement, the amount of coal which must be left varies with local conditions. It is difficult, if not impossible, to determine this factor in advance, and in attempting to approach as closely as possible the limit of safety, it often occurs that too much coal is removed. Even if the limit is not passed so far as immediate movement is concerned, it may be passed with reference to ultimate movement and the crushing of the pillars.

Apparently, so far as a large part of the State of Illinois is con-

* Gullachsen, Berent Conrad, "The Working of the Thick Coal Seams in Upper Silesia," Trans. Inst. Min. Engrs. Vol. 42, p. 209, 1911.

cerned, it is necessary to leave in the ground about one-half of the area of the coal if movement of the overlying beds is to be prevented.

The desirable dimensions of the rooms and the pillars vary widely from wide pillars between wide rooms to narrow pillars between narrow rooms. One company had squeezes when it drove 25-foot rooms on 50-foot centers, but it has had no trouble with 30-foot rooms with 60-foot centers. This is a question not simply of the crushing strength of the coal nor of the ability of the bottom to withstand pressure, but of the effect on the pillars of scaling at the sides. In other words, the strength of the pillar is not determined merely by its original size but by its effective size after the scaling action, which may follow the extraction of the room coal, has occurred. This scaling action is increased by the shattering effect of explosives.

The use of coal in the ground to prevent squeezes and subsidence, which is what abandoning of pillar coal amounts to, ought to be considered only as a last resort. It has been found that squeezes can be prevented by the removal of so little coal on the advance as to leave a solid support, and by the complete removal on the retreat so that the roof is left entirely without support. This process prevents the gradual settling which occurs when some support is left and produces a sharp bending, or localization of stress, sufficient to cause a rupture of the overlying rock and prevent the transference of weight from the mined-out area to the standing coal. This is the only certain method which has been found for the prevention of squeezes unless an absurdly large quantity of coal is abandoned. The means by which squeezes may be prevented vary under different conditions, but the essential consideration is that the roof of the mined-out area shall be left absolutely without support either from coal or from timber, so that it must fall.

TABLE 1¹
 DIMENSIONS OF ROOMS AND OF ROOM PILLARS AND PERCENTAGE OF
 EXTRACTION

LOCALITY AND COAL SEAM	Width of Room Feet	Width of Room Pillars Feet	Per Cent of Coal Left in First Working	Total Per Cent of Coal Recovered
Alabama:				
Newcastle seam.....	25	10-20	40	85
Thin seam.....	20	10-20	30	90
Blue Creek.....	25	25	35	85
Blount.....	25	25	30	80
Flat Top.....	25	25	35	85
Arkansas:				
Sebastian County.....	18-30, usually 24	12	30	70
Colorado:				
Trinidad series.....	20-25	20-25	50	50-80
Illinois:				
Springfield.....	30	15-20	60	60
Staunton (machine mines).....	30	30	60	60
Indian Territory.....	18-30, usually 24	10-12	30	70
Iowa.....	21-30	9-15	35	90
Maryland:				
Georges Creek.....	12-16	40-100	75	90
Pennsylvania:				
Connellsville.....	12	42	60	95
Connellsville.....	12	84	80	90
Pittsburgh.....	21-24	12-18	40	85-95
Clearfield.....	21-24	15	40-50	80-95
West Virginia:				
Fairmont.....	18-24	18-40	40-60	95
Clarksburg, Pittsburg seam.....	18	32	64	90
Mineral County, Pittsburg seam.....	12	48	80	95 (?)
Tucker County, Davis or Kittanning seam.....	18	22	35	90 (?)
Putnam County, Pittsburg seam.....	30	15	34	75-85
New River, Sewell-Nuttall seam.....	24	26	60	60-80
Thacker, (Thacker) Lower Kittanning seam.....	18-21	42	67	80
Logan, Lower Kittanning seam.....	21	20	57	
Kanawha (No. 2 Gas), Lower Kittanning seam.....	28	12-20	40	
Pocahontas, Double Entry, No. 3 Pocahontas seam.....	18	42	68	80
Pocahontas, Panel System, No. 3 Pocahontas seam.....	18	42		95 ²
Raleigh, Sewell-Nuttall seam.....	25	25	50	90

¹H. H. Stock, Mines and Minerals, Vol. 26, p. 107, 1905, and International Library of Technology, Vol. 150, par. 40, p. 60.

²Estimated.

The widths of rooms and room-pillars given in Table 1 show that there were few cases in which plans were made for the extraction of more coal from pillars than from the advance workings.

TABLE 21
PRINCIPAL FACTORS GOVERNING RECOVERY OF COAL IN DIFFERENT DISTRICTS

Operating District and State Item	Per Cent of Recovery Entire Seam	Ultimate Recovery Compared to Present	Period of Operation Years	Average Height of Seam, Feet	Roof Coal Carried	Nature of Top	Nature of Bottom	Pillar-Drawings	Clay Vains Encountered
1 Colorado, Southern.....	80-90	Same	5-30	8.5	24 in.	Slate.....	Soft slate	Yes	Dikes
2 Colorado, Other districts....	60-65	Same	5-30	8.5	18-24 in.	Very soft.....	Soft slate	Yes	None
3 Colorado, Other districts....	75-80	Better	10-35	3-7	Few places	Sandstone, poor shale....	Same as top	Yes	None
4 Michigan, Saginaw District..	65	Better	15	3	None	Black slate.....	Fire clay	Where allowed	None
5 Illinois, Central.....	80-90	Same	20	8	None	Slate, clod, and limestone	Fire clay	None	None
6 Illinois, Southern.....	65-70	Same	20	8	Yes	Sandy shale.....	Fire clay	To some extent	None
7 Illinois, Springfield District	55-75	Increase ²	20-25	6-7	Some places	Hard shale.....	Fire clay	Where allowed	None
8 Illinois, Williamson and Salline Counties ...	55-75	Increase ²	18	5.5	Some places	Hard shale.....	Fire clay	Where allowed	None
9 Illinois, Sherrard Field	90	Same	20	3.7	None	Blue rock and cap rock ..	Slate and sand rock	Yes	Yes
10 Pennsylvania, Extreme southwest section.....	72.5	Better	3	7.5	10 in.	1.5-3 ft. draw slate.....	Soft fire clay	Yes	None
11 Pennsylvania, Westmoreland Co..	84	Below	25-35	6.7	None	Slate.....	Fire clay	Yes	None
12 Pennsylvania, Somerset County ..	95	Same	8	3.9	None	Hard black slate.....	Limestone	Yes	Very few
13 Maryland, Georges Creek Field	97	Same	12	3	None	Sand rock.....	Sand rock	Yes	Yes
14 Maryland, Georges Creek Field	88	Same	94	9	18 in.	Gray shale coal, dark shale	Hard gray shale	Yes	None
15 Ohio, Belmont County ..	60	Same		5-6	None	Slate and shale.....	Fire clay	None	None
16 Ohio, Harrison County	70-75	Same		3.7-5	None	10 in. firm slate.....	Fire clay	None	None
17 West Virginia.....	90	Same	50	8	Some places	Varies.....	Fire clay	Yes	Yes
18 Alabama, no reply....									
19 Tennessee, no reply....									
20 Kansas, no reply....									
21 Iowa, no reply....									
22 Kentucky, no reply..									

1A. W. Hesse, "Maximum Coal Recovery," W. Va. Min. Inst., June 3, 1914; Coll. Eng., Vol. 35, p. 13; and Coal Age, Vol. 5, p. 1051.
² Operated by panel method, all others by room-and-pillar.

TABLE 3
DIMENSIONS OF WORKINGS AND ESTIMATED PERCENTAGES OF EXTRACTION IN ILLINOIS MINES

District	No. of Coal Bed	No. of mine (Coop. Investigations)	Depth of hoisting shaft	Method		Entry width			Entry pillar width			Barrier pillar width		Room		Width of room pillar	Room neck		Dist. from entry to full room width	Dist. between room centers	Width of room stump	Width of cross-cuts	Has mine had squeezer?	Operators' Estimate of Percent- age Extracted	
				Unmodified Room-and-Pillar	Panel	Main	Cross	Room	Main	Cross	Room	Main	Cross	No. rooms on room entry	Width		Length	Width							Length
II	2	12	125	×	×	8	8	—	12	12	23	—	—	16	24	150	18	8	10	20	42	34	8	No	44
	2	13	114	×	×	8	8	—	20	20	30	30	—	—	24	250	20	8	4	20	44	35	9	No	55
	2	14	135	×	×	8	8	—	18	18	20	—	—	—	25	250	22	18	12	22	50	42	8	No	46
	—	—	—	×	×	8	8	—	18	18	20	—	—	—	25	240	17	8	10	20	42	34	10	No	49
	—	—	—	×	×	8	8	—	—	—	—	—	—	—	—	—	—	—	—	—	—	—	—	—	—
III	1	17	210	×	×	8	8	—	30	30	50	—	—	—	20	250	20	8	7	14	40	32	12	No	96
	1	18	69	×	×	12	12	—	42	15	36	—	—	—	26	250	19	8	7	12	45	37	12	Yes	96
	1	19	70	×	×	12	12	—	18	18	18	—	—	—	24	250	18	8	7	25	42	34	8	No	50
	2	22	60	×	×	6	6	—	—	—	15	—	—	—	15	250	15	6	15	23	30	24	15	Yes	45
	1	24	40	×	×	12	12	—	20	20	15	—	—	—	24	300	21	8	9	29	45	37	15	No	70
IV	—	26	196	×	×	8	8	—	30	21	50	—	—	—	26	190	6	8	8	30	32	24	8	No	56
	—	29	185	×	×	8	8	—	36	24	40	—	—	—	24	180	8	8	7	18	32	24	8	No	57
	—	30	60	×	×	10	8	—	15	12	25	—	—	—	22	210	10	8	7	18	32	24	13	No	57
	—	34	200	×	×	8	10	—	30	34	38	—	—	—	28	180	10	10	9	30	38	28	4	Yes	42
	—	41	365	×	×	16	16	—	20	30	80	—	—	—	25	200	10	12	9	15	35	23	8	No	56
	—	25	185	×	×	8	8	—	36	30	24	—	24	23	240	180	6	8	7	18	30	22	8	Yes	54
	—	27	170	×	×	12	12	—	20	15	50	—	—	—	30	200	8	8	9	20	38	30	8	No	53
	5	28	150	×	×	8	8	—	25	20	40	—	—	—	20	180	10	8	10	15	34	26	8	Yes	54
	—	32	68	×	×	8	8	—	25	20	40	—	—	—	20	150	12	8	10	15	32	24	8	No	57
	—	33	285	×	×	8	8	—	30	30	50	—	—	—	30	200	10	8	10	35	36	28	6	No	64
	—	35	187	×	×	8	8	—	35	30	60	—	—	—	26	240	9	8	9	27	35	27	12	Yes	56
	—	36	238	×	×	12	12	—	30	24	40	—	—	—	24	250	10	8	9	27	35	27	12	Yes	48
	—	37	235	×	×	10	10	—	30	30	60	—	—	—	25	300	12	10	15	15	42	30	10	Yes	41
	—	38	245	×	×	10	10	—	27	24	35	—	—	—	30	240	12	12	15	15	42	30	10	No	62
	—	39	204	×	×	12	12	—	30	30	30	—	—	—	30	250	12	12	11	18	42	30	10	Yes	65
	—	40	270	×	×	12	12	—	30	24	35	—	—	—	24	200	6	8	9	18	30	22	7	Yes	50
V	—	43	160	×	×	14	14	—	30	30	40	—	—	—	30	250	20	18	10	20	50	32	18	Yes	65
	—	44	270	×	×	15	18	—	25	25	50	—	—	—	22	250	24	18	12	28	46	28	18	Yes	59
	—	45	320	×	×	16	16	—	24	20	—	—	—	—	27	250	20	19	12	30	47	28	15	No	64
	5	46	450	×	×	14	14	—	20	20	50	—	—	—	33	200	7	19	12	25	40	21	19	Yes	82
	—	47	90	×	×	14	19	—	20	12	20	—	—	—	22	250	12	8	6	13	34	26	12	No	68
	—	48	76½	×	×	14	14	—	25	25	50	—	—	—	24	300	21	12	6	15	45	33	12	No	59
	—	49	337	×	×	14	14	—	30	21	35	—	—	—	24	250	9	12	9	25	33	21	12	Yes	73

TABLE 3 (Continued)
DIMENSIONS OF WORKINGS AND ESTIMATED PERCENTAGES OF EXTRACTION IN ILLINOIS MINES

District	No. of Coal Bed	No. of mine (Coop. Investigations)	Depth of hoisting shaft	Method		Entry width			Barrier Pillar width		No. rooms on room entry	Room			Room neck	Dist. from entry to full room width	Dist. between room centers	Width of room stump	Width of cross-cuts	Has mine had squeezer?	Operators' Estimate of Percentage Extracted		
				Unmodified Room-and-Pillar	Panel	Entry width		Barrier Pillar width		Width		Length	Width of room pillar										
						Main	Cross	Main	Cross														
VI		50	726	×	—	12	12	—	40	100	—	35	250	22	25	12	24	34	47	35	22	Yes	58
		55	160	×	—	10	9	—	50	50	—	40	250	25	26	15	19	51	36	15	Yes	46	
		56	580	×	—	12	12	—	30	100	—	40	250	26	26	12	23	45	52	40	No	50	
		59	200	×	—	9	9	—	20	50	—	20	225	23	23	15	16	32	38	26	15	Yes	68
		60	220	×	—	12	12	—	18	20	—	20	300	21	22	12	6	24	33	21	Yes	51	
		62	190	×	—	10	10	—	30	75	—	30	250	20	20	15	15	35	25	10	Yes	51	
		63	140	×	—	10	10	—	30	40	—	30	500	18	10	12	18	40	30	12	Yes	64	
		64	120	×	—	11	11	—	19	38	—	22	300	22	20	15	15	35	26	8	No	58	
		65	90	×	—	12	12	—	20	10	—	20	250	21	24	9	6	24	30	20	Yes	58	
		51	494	—	×	9	9	—	30	30	100	100	250	16	16	24	18	38	40	31	18	Yes	52
		52	640	—	×	12	12	—	50	50	38	150	250	25	12	18	41	50	38	12	No	58	
		53	450	—	×	16	16	—	28	50	50	100	300	11	20	18	40	22	34	7	Yes	65	
		54	380	—	×	15	7	—	50	30	150	20	250	25	12	10	32	50	38	18	No	55	
		57	320	—	×	12	12	—	50	38	100	100	20	25	300	25	10	32	50	38	18	No	55
		58	516	—	×	12	12	—	25	20	20	125	200	22	12	15	34	34	22	12	Yes	56	
VII		68	387	×	—	16	16	—	40	40	—	27	235	32	33	9	15	27	65	56	20	Yes	50
		70	92	×	—	14	14	—	50	50	—	50	300	30	30	21	25	50	39	21	No	50	
		72	287	×	—	21	21	—	40	40	—	55	300	30	30	21	6	60	39	21	No	56	
		78	160	×	—	21	21	—	60	30	—	60	300	30	30	21	24	73	52	21	Yes	62	
		79	127	×	—	21	21	—	60	30	—	60	250	30	250	21	18	38	60	39	21	No	59
		80	145	×	—	12	14	—	18	35	—	60	35	250	45	25	40	80	59	21	Yes	55	
		81	200	×	—	21	21	—	80	40	—	60	40	250	40	21	18	36	80	59	30	Yes	63
		82	192	×	—	14	21	—	42	42	—	60	30	300	21	15	25	60	39	21	No	60	
		84	320	×	—	21	21	—	39	39	—	150	300	30	30	21	25	60	39	21	No	63	
		87	707	×	—	12	12	—	40	40	—	40	250	29	250	12	20	50	60	48	20	No	58
		88	85	×	—	12	12	—	20	20	—	40	250	29	250	17	12	6	46	34	12	Yes	65
		89	85	×	—	8	8	—	20	15	—	40	250	29	250	10	24	45	37	8	Yes	48	
		90	160	×	—	21	9	—	60	30	—	110	33	250	17	9	12	24	50	41	No	63	
		66	332	×	×	21	21	—	40	40	75	30	300	30	30	21	25	60	39	21	Yes	56	
		67	320	×	×	12	12	—	30	30	27	60	300	20	12	6	30	45	33	12	No	47	
		69	290	×	×	10	10	—	40	40	60	60	27	23	200	9	18	36	65	56	10	Yes	51
		71	194	×	×	21	21	—	60	40	13	60	300	27	9	12	30	50	29	21	No	60	
		73	318	×	×	12	12	—	60	60	12	50	300	20	21	12	30	60	48	12	Yes	50	
		74	330	×	×	12	12	—	50	40	25	50	250	27	10	12	30	48	36	12	Yes	52	
		75	310	×	×	12	12	—	50	40	75	75	28	225	27	10	20	35	55	33	21	Yes	44
		76	370	×	×	10	10	—	50	40	40	100	300	25	22	28	55	33	21	Yes	63		
		77	462	×	×	14	14	—	50	50	40	35	250	19	9	12	24	70	49	21	Yes	50	
		83	140	×	×	21	21	—	35	35	12	35	250	25	35	25	44	35	12	Yes	55		
		85	440	×	×	12	12	—	50	25	50	50	250	19	9	25	44	35	12	Yes	55		
		86	536	×	×	12	8	—	100	50	16	100	400	25	12	18	30	50	32	8	No	52	

TABLE 3 (Concluded)

DIMENSIONS OF WORKINGS AND ESTIMATED PERCENTAGES OF EXTRACTION IN ILLINOIS MINES

District	No. of Coal Bed	No. of mine (Corp. Investigations)	Depth of hoisting shaft	Method		Entry width				Entry Pillar width			Barrier Pillar width		No. rooms on room entry	Room			Room neck		Dist. from entry to full room width	Dist. between room centers	Width of room stump	Width of cross-cuts	Has mine had squeezes?	Operators' Estimate of Percentage Extracted	
				Unmodified Room-and-Pillar	Panel	Main	Cross	Room	Main	Cross	Room	Main	Cross	Width		Length	Width of room pillar	Width	Length								
VIII	6	91	217	×	×	9	9	—	25	21	—	40	—	—	—	43	200	4	9	19	18	47	38	5	Yes	55	
	6	92	240	×	×	7	7	—	35	30	—	60	—	—	—	24	240	3	9	9	18	27	18	9	Yes	68	
	6	93	186	×	×	8	9	—	21	21	—	50	—	—	—	24	200	16	9	9	9	40	31	9	Yes	75	
	7	94	90	×	×	7	6	—	16	12	—	17	—	—	—	24	240	6	9	9	9	30	21	9	No	81	
	6	95	90	×	×	6	6	—	21	21	—	50	—	—	—	25	500	11	9	12	12	36	27	9	No	82	
	7	96	223	×	×	7	8	—	21	21	—	50	—	—	—	21	150	9	9	9	12	30	21	7	No	68	
	7	97	223	×	×	7	8	—	21	21	—	50	—	—	—	21	150	9	9	9	12	30	21	7	No	68	
Averages by districts																											
II	2		133	—	—	8	8	—	17	17	—	25	—	—	—	25	222	19	8	9	21	45	36	9	—	48	
III	1 & 2		90	×	—	10	10	—	28	21	—	30	—	—	—	22	260	18	8	9	21	40	33	12	—	71	
IV	5		202	—	—	10	10	—	27	25	—	44	—	—	—	26	197	9	9	9	21	35	26	9	—	55	
V	5		243	×	—	15	16	—	25	22	—	41	—	—	—	26	250	16	15	10	21	42	27	15	—	67	
VI	6		348	—	—	11	11	—	33	33	—	81	—	—	—	23	262	18	11	14	28	41	30	13	—	56	
VII	6		279	—	—	16	15	—	46	39	—	60	—	—	—	31	277	28	16	16	32	58	42	17	—	56	
VIII	6 & 7		174	×	—	7	7	—	24	21	—	43	—	—	—	27	255	8	9	13	35	26	8	—	71		
Aver. of 30 panel mines			306	—	×	12	12	12	39	35	25	68	56	23	27	251	18	12	13	24	45	33	11	...	...	55	
Aver. of 48 room-and-pillar-mines.....			208	×	—	12	12	..	31	27	..	46	..	..	26	250	19	12	12	24	45	33	13	...	...	54	

TABLE 4¹
VALUES² OF SURFACE AND OF COAL RIGHTS BY COUNTIES IN ILLINOIS

County	Value of Coal per Acre	Number of Coal Bed	Average Surface Value, Census of 1910
Bond.....	\$ 25	6	\$ 45.43
Bureau.....	10-100	2	114.53
Christian.....	10- 50	6	123.63
Franklin.....	35-100	6	38.48
Fulton.....	15-100	5	88.18
Gallatin.....	20- 25	5	48.60
Grundy.....	10- 25	2	75.52
Henry.....	135	6	112.03
Jackson.....	25- 75	2, 6	31.27
La Salle.....	10-100	2, 5	142.92
Livingston.....	10- 50	6	161.76
Logan.....	20- 50	5	156.49
Macoupin.....	15- 50	6	69.74
Madison.....	10- 40	6	70.53
Marion.....	20	6	39.45
Marshall.....	15	2	123.92
McLean.....	15	5	171.85
Menard.....	25- 30	6	122.04
Montgomery.....	25- 50	6	73.49
Morgan.....	20- 30	6	124.28
Peoria.....	20- 50	5	107.67
Perry.....	25	6	30.62
Putnam.....	15	2	104.69
Randolph.....	25	6	36.11
St. Clair.....	10-100	6	81.57
Saline.....	50-150	5	39.88
Sangamon.....	20-100	5, 6	138.30
Scott.....	10- 40	2	83.21
Shelby.....	10- 25	6, 5	88.72
Vermilion.....	100-150	6, 7	138.85
Warren.....	15	1, 2	129.80
Will.....	15	2	104.08
Williamson.....	50-150	6	30.61
Woodford.....	15	2	154.27

¹Young, Lewis E., "Surface Subsidence in Illinois," Ill. Coal Min. Invest., Bulletin 17, p. 55, 1916.

²These prices are not offered as an authoritative basis for valuation but indicate in a general manner the prices at which coal has been sold or at which it is held in some of the important counties.

CHAPTER II

EXTRACTION IN ILLINOIS

8. *Plan for Division of State into Districts.*—At the beginning of the work of the Coöperative Coal Mining Investigations, the State was divided into districts in order that those beds which are similar in general conditions might be studied and considered together. This subdivision into districts is shown by Fig. 2, and the districts are described in Table 5.

TABLE 5
DISTRICTS INTO WHICH THE STATE HAS BEEN DIVIDED FOR THE PURPOSES
OF INVESTIGATION

Investi- gations, District	Coal Seam	Method of Mining	Counties	Investi- gations Numbers for Mines Examined
I	2	Long-wall	Bureau, Grundy, La Salle, Marshall, Putnam, Will, Woodford.....	1 to 11
II	2	Room-and-pillar	Jackson.....	12 to 16
III	1 and 2	Room-and-pillar	Brown, Calhoun, Cass, Fulton, Greene, Hancock, Henry, Jersey, Knox, McDonough, Mercer, Mor- gan, Rock Island, Schuyler, Scott, Warren.....	17 to 24
IV	5	Room-and-pillar	Cass, DeWitt, Fulton, Knox, Logan, Macon, Mason, McLean, Menard, Peoria, Sangamon, Schuyler, Taze- well, Woodford.....	25 to 42
V	5	Room-and-pillar	Gallatin, Saline.....	43 to 49
VI	6 (east of Duquoin anticline)	Room-and-pillar	Franklin, Jackson, Perry, William- son.....	50 to 65
VII	6 (west of Duquoin anticline)	Room-and-pillar	Bond, Christian, Clinton, Ma- coupin, Madison, Marion, Mont- gomery, Moultrie, Perry, Randolph, Sangamon, Shelby, St. Clair, Wash- ington.....	66 to 90
VIII	6 and 7 (Danville)	Room-and-pillar	Edgar, Vermilion.....	91 to 97

In the present publication the conditions prevailing and the methods followed in the various districts are described, and the extent to which these affect the percentage of recovery is discussed. Material and information has been gathered at various times, and

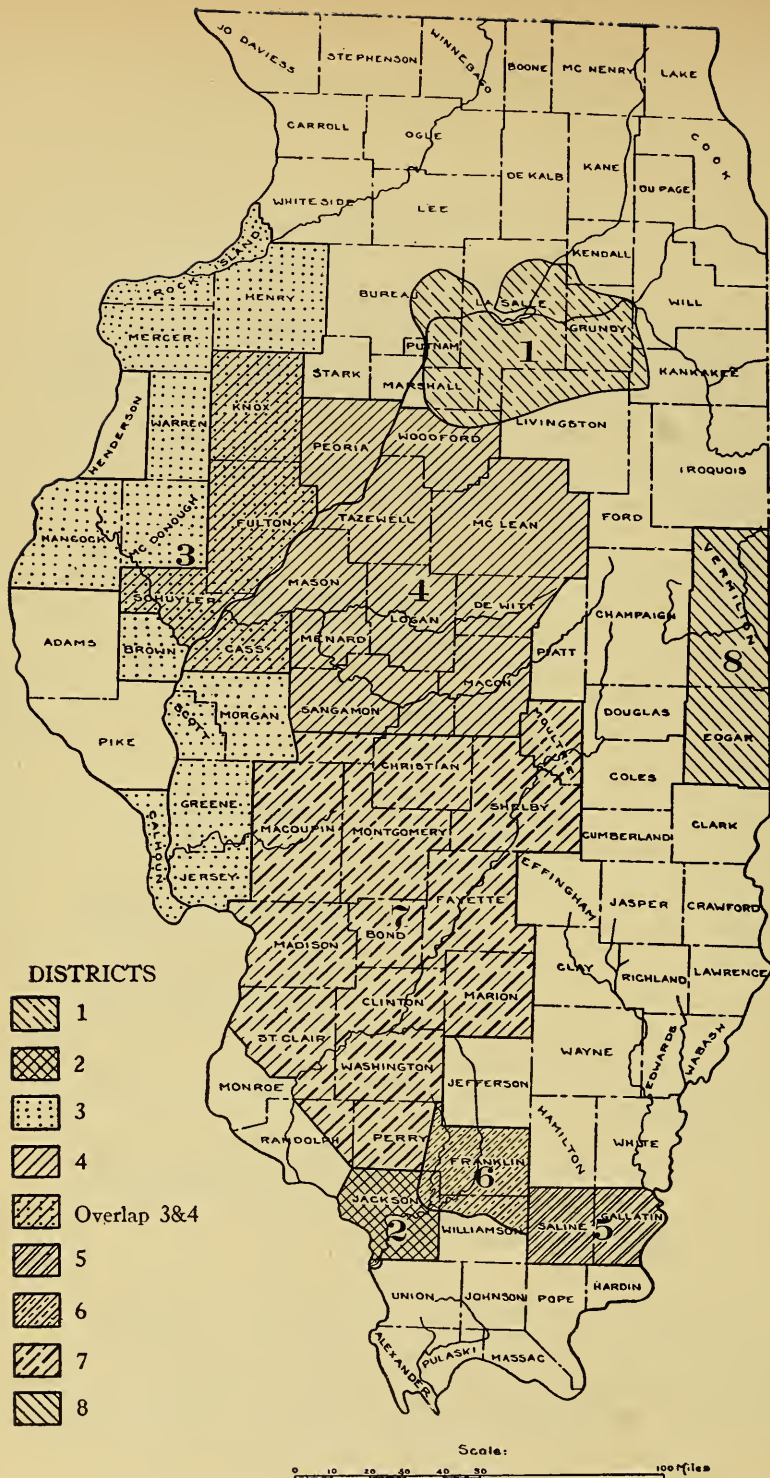


FIG. 2. MAP OF DISTRICTS OF COÖPERATIVE COAL MINING INVESTIGATIONS

some of it, especially that relating to physical conditions and usual methods of operation, has been published in previous bulletins of this series. These facts are summarized in Bulletin 13.*

9. *Conditions Affecting Extraction.*—Since there is an immense quantity of coal underlying the state and only a comparatively small portion has been extracted, it is perhaps natural that little serious attention has been given to the subject of high recovery. Those controlling production have been concerned principally with other phases of the subject, not because they have been indifferent to the highest possible utilization of resources, but because they have believed that the methods in use were giving the lowest possible cost of production; and low cost of production has been regarded as a necessity for the development of the Illinois fields in competition with other coal fields.

Table 3, rearranged from Bulletin 13, gives the dimensions of the workings and the estimates of recovery for the mines examined by the Coöperative Coal Mining Investigations.

The values for the percentage of extraction given in the last column of Table 3 are, in most cases, founded upon estimates furnished by the operators. In many instances subsequent investigation has shown that these values are not correct. There are only a few mines in the state from which it has been possible to obtain accurate data on recovery because of the lack of information on which such data could be based. Generally, it has been found that persons estimating the percentage of recovery have been inclined to use values too high and have failed to take into account some of the sources of loss. Later figures on extraction, the most trustworthy it has been possible to obtain, will be found in the descriptions of the districts.

10. *District I.*—The No. 2 bed varies in thickness from two feet, eight inches to four feet, the average thickness being about three feet, two inches. On the east side of the LaSalle anticline the thickness of cover ranges from 40 to 200 feet; on the west side the bed lies at a depth of 350 to 550 feet. In the eastern, or Wilmington, section the roof is a smooth gray shale, though sandstone is found in some places. In the western or LaSalle field the roof is a gray

* Andros, S. O., "Coal Mining in Illinois," Ill. Coal Min. Invest., Bul. 13, 1915.

shale. In the Wilmington field the floor is a dark gray fire clay varying in thickness from a few inches to several feet. When this clay is wet, it heaves badly under pressure. In the LaSalle field the floor is fire clay, but a hard sandstone is sometimes found immediately beneath the coal.*

Nearly all the coal produced in this district is mined by the long-wall method, and this method, of course, gives the highest possible percentage of recovery. G. S. Rice says that at one mine in which a record was kept for six years the loss of coal from all causes was five per cent.†

11. *District II.*—The No. 2 seam is found under shallow cover ranging from 25 to 160 feet. In most places the floor is sandstone, but shale or clay is occasionally found. In places a wet and fluid sand is found about thirty feet below the surface, and it has a marked effect upon surface subsidence, causing the formation of rather deep pits instead of gentle sags. The bed is divided into two benches by a shale parting, varying in thickness from one-eighth inch to thirty-six feet. The bottom bench varies in thickness from $3\frac{1}{2}$ to 4 feet, and the top bench has an average thickness of two feet. Where the parting between the benches is less than four inches thick, the two benches of the seam are worked as one and the working faces in rooms and entries are from six to seven feet high. Where the parting is more than four inches thick, only the lower bench is mined and the parting becomes the mine roof. When both benches are worked and the bed is more than six feet thick, only the lower six feet of coal are mined, eight to twelve inches of top coal being left; but if the coal is not more than six feet thick the full thickness of the bed is mined, and the gray shale overlying the coal becomes the roof.

With one exception, the mines examined are operated by the unmodified room-and-pillar method. Operations are carried on without close adherence to the projected sizes of rooms and pillars. The result of this practice is a rather high percentage of extraction, as pillars are gouged to a considerable extent.‡ At one mine in this

* Andros, S. O., "Coal Mining Practice in District I," Ill. Coal Min. Invest., Bul. 5, p. 10, 1914.

† Rice, George S., "Mining-Wastes and Mining-Costs in Illinois," Trans. Amer. Inst. Min. Engrs., Vol. 40, p. 31, 1909.

‡ Andros, S. O., "Coal Mining Practice in District II," Ill. Coal Min. Invest., Bul. 7, p. 9, 1914.

district which is operated on the panel system and in which a serious attempt is made to remove pillars as far as possible, the percentage of extraction is probably higher than at any other mine in southern Illinois. At this mine the shaft is 115 feet deep. There are triple main and cross entries, each ten feet wide, with 20-foot entry pillars. Barrier pillars on main and cross entries are twenty feet wide. Rooms are twenty feet wide with 10-foot pillars. All cross-cuts are eight feet wide. Although there are no exact figures on the percentage of recovery, it is evident from the dimensions of the workings that about two-thirds of the coal is extracted in the first working. Since by slabbing pillars, forty to fifty per cent of the pillar coal is also obtained, the final recovery probably amounts to about eighty per cent. The rooms are widened about thirty feet before the end is reached, little or no pillar coal is left beyond this point, and as much of the remainder of the pillars as possible is taken out by slabbing.

The possibility of extracting a large amount of pillar coal depends upon the character of the top which may be allowed to fall without serious consequences, because the shale and sand overlying the coal seal the opening so that the influx of water is not seriously increased by a break. When the top falls, the necks of the rooms are boarded up and the water is handled by a pump.

12. *District III.*—The No. 1 and No. 2 beds are worked. The cover overlying the coal is thin. The topography of the surface in many places is rolling, with hills about 150 feet high near Mather-ville. Bed No. 2 lies at depths of seven feet to one hundred feet with an average cover of fifty-five feet. Bed No. 1 averages four feet in thickness and is broken in places by small faults, slips, clay veins, and rolls. A poorly developed parting divides the bed into two benches, the upper of which is in most places about two feet thick.

The immediate roof in the northwestern part of the district is of hard black shale which is easy to support. In the southern part a bituminous calcareous shale, two to five inches thick, lies in places immediately over the coal. This shale, called clod, is hard when first exposed to the air but after exposure softens and falls. Throughout the district the cap rock is limestone. In limited areas where the shale is missing, this limestone forms the immediate roof. Above

the cap rock occurs a dense, fine-grained, non-crystalline limestone locally called "blue rock."

Below bed No. 1 there occurs in places an irregular band of hard bone, three to six inches thick. The floor proper is of light gray micaceous fire clay which contains plant stems and roots. This clay heaves badly when wet and sometimes swells enough to fill the entry. In parts of some mines a carbonaceous shale lies between the fire clay floor and the coal; sometimes this shale is supplanted by sandstone. These casual deposits are called "false bottoms."

Bed No. 2 varies in thickness from 1 foot, 10 inches to 4 feet, and averages 2 feet, 6 inches. The bed has a slight dip to the east. A band of mother coal and iron pyrites persists throughout the bed. This occurs about fourteen inches from the roof. The immediate roof is of smooth and regular calcareous shale, known locally as soapstone. The floor is of soft gray fire clay which contains nodular concretions of iron pyrites called sulphur balls. The coal in this district lies near the surface, but at no point is the overburden stripped.

Except at two mines, the mining system is the simplest form of double-entry room-and-pillar. Table 3 shows the dimensions of workings in the mines examined. The coal is gained during the first working with a waste of pillar coal amounting to about 45 per cent of the bed. At the two exceptions 75 per cent of the pillar coal is recovered on the retreat, a large percentage for Illinois room-and-pillar mines.

A main entry and a parallel air-course, each six feet high and eight feet wide, are driven from each side of the shaft toward the boundaries. At right angles to these main entries, pairs of cross entries are driven every 500 feet. On the cross entries, after leaving a barrier pillar of 50 feet, rooms are turned on 45-foot centers. Room necks are 7 feet long and 8 feet wide, and are widened to the left at angles of about 45 degrees; thus they reach the full room width of 26 feet at distance of 14 feet from the beginning of the widening. After the first room on each entry has been holed through, the room-pillar cross-cuts are closed by gob stoppings, and the line of No. 1 rooms is kept open; thus two additional air-courses are provided.

After the entry has been driven to the limit and the rooms on it have been worked out, the last pillar on the entry is drawn; then the other room pillars are drawn until the pillar between rooms

3 and 4 is reached. The room pillars between the main entry and room 4 are left to protect the main entry and air-course. The method of drawing pillars is illustrated in Fig. 3. When the room is driven

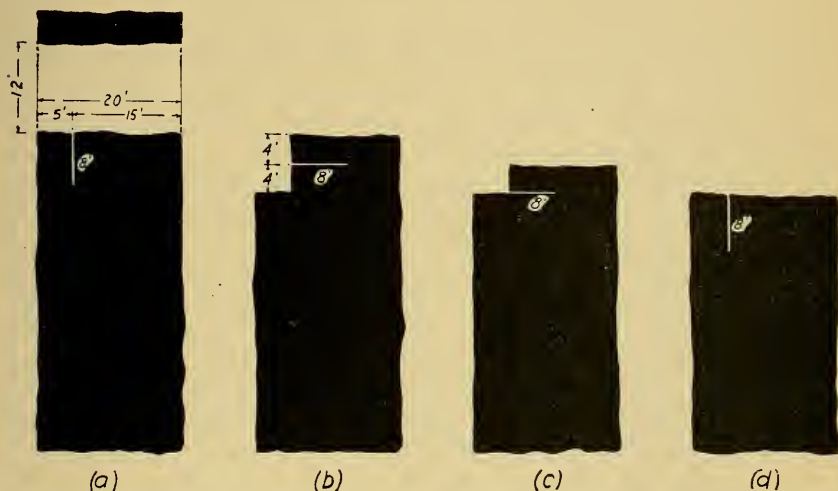


FIG. 3. PILLAR DRAWING AT MATHERVILLE, ILLINOIS

up to its full length, a 12-foot cut is made across the end of the pillar (a), a 5-foot slab about 8 feet long is shot from the side of the pillar, a 4-foot slab is shot from the end (b), and the end of the pillar is squared up by shooting off another 4-foot slab (c). Beginning again at (d), the process is repeated.

The hard roof is easy to support and often stands while 25 to 200 feet of pillars are being drawn. When the weight of the roof becomes too heavy, the roof breaks at the pillar ends. The cracking of the props gives ample warning of the break, and work is discontinued until the roof falls. The interval between the first heavy cracking of props and the breaking of the roof is usually not more than twelve hours.

A break line of about twenty-five degrees with the face of the rooms is roughly maintained. When roof falls prevent access to the squared-up pillar ends, a 12-foot cut is again made completely through the pillar, as at the face of the room when drawing began, and with this new pillar end the procedure continues; consequently, very little pillar coal is lost. Carl Scholz, President of the Coal

Valley Mining Company, states that at mine No. 3 at Matherville the loss of pillar coal does not exceed four per cent.

At the No. 3 mine of the Coal Valley Mining Company, the cost of producing coal is much less on pillars than on advance work in rooms. Room coal costs on the average \$1.25 per ton at the pit mouth, and pillar coal costs \$1.015. This difference in cost exists because track, yardage, bottom digging, and driving through rolls and slips are properly charged against room coal, while there are no such charges against pillar coal. When pillars are drawn, therefore, the average cost per ton for the total production is materially reduced. At this mine rooms are worked with one man at the face, but two men are placed at each pillar and at the face of each entry. Only one man has been injured in connection with the pillar drawing.

With the extraction of such a large percentage of the bed surface subsidence is to be expected. The topography of the surface is rolling, and subsidence is usually indicated by cracks in the hill-sides. The largest single area affected was reported to be one acre which subsided from 6 to 12 inches.*

13. *District IV.*—In District IV the No. 5 coal is mined. The average thickness of this coal is 4 feet, 8 inches according to data taken at 240 mines and given in the Thirty-first Annual Coal Report of Illinois. The No. 5 bed outcrops in Peoria, Fulton, and Knox Counties, but is found at greater depths toward the east. It lies from 300 to 600 feet below the surface in Macon County, 400 feet in McLean, and from 260 to 300 feet in Logan.

The roof is of black sheety shale varying in thickness from a few inches to 35 feet and containing occasionally "niggerheads" of pyrite. In many mines there is, in places, a layer of pyrite two or three inches thick between the coal and the shale. Where this layer is present, the shale is protected from the air and stays up; where it is not present, the shale falls badly and sometimes caves to a height of 35 feet. A limestone occurs above the shale in most mines, though in a few places a fine grained micaceous sandstone is found. In some cases the shale is absent, and the cap rock becomes the roof.

A great many clay veins extend through the coal and the roof shale; there are also small faults, slips, and rolls, and places where

* Andros, S. O., "Coal Mining Practice in District III," Ill. Coal Min. Invest., Bul. 9, pp. 11 et seq., 1915; "Coal Mining in Illinois," Ill. Coal Min. Invest., Bul. 13, 1915.

the coal has been eroded and the space has been filled with drift. It is difficult, therefore, to calculate the total tonnage and to project any plan of operation. In many places the coal adheres to the roof and separates from it with difficulty. In one mine about an inch of coal is left to protect the roof shale from the air. In most mines the floor consists of a dark gray clay which heaves badly when wet.

Operations are conducted on the unmodified room-and-pillar system or on the so-called panel system. Dimensions of workings are given in Table 3. There are also four mines in the district which are operated on the long-wall system. Mining methods have not been given very careful attention, and the variations in the coal bed tend to minimize the effect of such attention as has been given. The method of mining generally practiced in the district involves the running of parallel main entries from the shaft toward the boundaries, and the turning of cross entries from the main entries at intervals of 350 to 400 feet. Rooms are turned off these cross entries on 30-foot to 42-foot centers, and are driven 20 to 30 feet wide. Room pillars average 9 feet in width and rooms 26 feet, but pillars are gouged as the miner pleases. This haphazard method is productive of so many squeezes that in some mines a modification of the system has been employed in which stub or room entries are turned off the cross entries. This method approaches the panel system and is called, locally, "block-room-and-pillar." Sometimes a sufficiently large cross barrier pillar is left to confine a squeeze to the block in which it originates, but generally the barrier pillar is gouged and squeezes ride over it unchecked until they reach a horseback or some ungouged pillar which is large enough to stop them. In several mines squeezes originating in rooms have traveled to the main barrier pillar and to the solid coal at the entry face. In one mine an entry was saved from a threatened squeeze by very heavy timbering ahead of the squeeze.

Eleven of the sixteen mines examined are at present operated on this semi-panel system, but the relative dimensions of room and room pillar have not been changed from previous operations. These dimensions are not safe under the roof found in the district. Room width is not uniform, but rooms are narrowed to avoid horsebacks and widened again where the coal resumes its normal thickness. There is a temptation to get all the coal possible on the advance,

because the numerous rolls make uncertain the total tonnage which can be extracted from any area, and the rolls interfere seriously with any projected plan since cutting through them is expensive.

Pillars are drawn in only a few mines, and in these drawing is not done systematically but is confined to shooting slabs off the thickest parts of the pillars. Room pillars are tapered to cross-cuts in nearly all mines. In one case an attempt was made to draw pillars, and a track was laid along the rib, but objections were raised by the miners to this position of the track, and the attempt was abandoned. Principally because of the insufficient pillar-width, the floor of fire clay heaves badly even when dry.*

Nineteen mines were examined in this district, and the estimates of the percentage of recovery furnished at seventeen ranged from 55 to 75 per cent, averaging 67.26 per cent. It is probable that most of the estimates are too high for, although the gouging of pillars tends toward high percentage of extraction, careless methods always result in the loss of much larger quantities of coal than is supposed. One company, which has given careful attention to the forms and dimensions of its workings, is extracting about 70 per cent of the coal. It is doubtful however, if the extraction throughout the district as a whole amounts to 60 per cent.

14. *District V.*—Bed No. 5 in Saline and Gallatin Counties lies at a depth of 25 to 450 feet, being nearest the surface along the southern portion of the district. The bed varies in thickness from 4 to 8 feet, and averages $5\frac{1}{3}$ feet in Saline County and 4 feet in Gallatin County.

The roof of the No. 5 coal in this district is of shale which is sometimes laminated and interbedded locally with bone and stringers of coal for a distance of 3 feet above the seam. The roof usually contains many concretions of iron pyrites called "niggerheads." It breaks quickly when wide spans are left supported, and it is drawn when it shows a plainly marked parting not more than 4 inches above the coal; but such a parting rarely occurs and the coal bed is so thin that the top coal cannot profitably be left in place. There are numerous falls which can be avoided only by making entries narrower than at present.

* Andros, S. O., "Coal Mining Practice in District IV," Ill. Coal Min. Invest., Bul. 12, pp. 15 and 19, 1915.

The floor is of fire clay which in places contains much sand and heaves badly when wet. The bed contains many hills and rolls causing grades as high as 15 per cent in the entries of some mines. The coal is not pinched out at these hills, but follows the contours with undiminished thickness. In some mines about 9 inches of bottom coal is left below a "blue band," but as this bottom coal is not of good quality, increased facility in shooting compensates for the loss of coal.

The room-and-pillar system of mining is used exclusively, a main haulage entry and a parallel air-course being noted in every mine examined except one,—in which triple main entries were driven, two for intake air and one for return air and haulage. In the smaller mines and in many of the larger ones, the dimensions of workings are not suited to the roof conditions. The main entries vary in width from 14 to 16 feet. A few shaft pillars have been gouged. The room stumps, which are left when rooms are turned off the cross entries, are generally small. The closing of entries by roof falls may often be attributed to local squeezes which ride over the room stumps. Table 3 gives dimensions of workings for each mine examined.

The custom of driving wide rooms and entries, of leaving narrow pillars throughout the mine; and of obtaining all the coal possible on the advance without attempting to draw pillars has resulted in a high percentage of extraction for Illinois mines. The percentages given in Table 3 were calculated from the most nearly exact data obtainable at the time of their publication but are unquestionably too high. This reported extraction, averaging 67.1 per cent for the seven mines examined, was accomplished only with greatly increased expense for cleaning up.* One of the large operators of this district reports an average recovery at ten mines of 60.5 per cent over a 5-year period with a maximum of 72 per cent and a minimum of 52 per cent where the cover varies from 60 to 414 feet. Pillar drawing is not practiced, and it would be impossible to gain the percentages of coal given in the table if the dimensions given were adhered to, but pillars are gouged to such an extent that there should be a higher percentage of extraction than is calculated from the dimensions of rooms and pillars in the table.

*Andros, S. O., "Coal Mining Practice in District V," Ill. Coal Min. Invest., Bul. 6, pp. 9 and 12, 1914.

15. *District VI.*—This district has experienced a rapid development, because the No. 6 coal commands a ready market; consequently mining on a large scale is possible. Bed No. 6 lies close to the surface along the Duquoin anticline* but dips sharply to the east, reaching a depth of 726 feet at Sesser. A general uplift has brought it to the surface along an east-west line extending through Carterville to Marion and along a southeast line from Marion to the boundary of the district. East of the area affected by the Duquoin anticline, the bed has a pronounced dip to the north. Along the outcrop line there are a few slopes and strippings, but the steep dip of the bed leaves only a small acreage with thin cover, and the remaining openings are shafts. The seam itself is thick, ranging from $7\frac{1}{2}$ to 14 feet and averaging, as shown by 130 borings, 9 feet, 5 inches. A clean persistent parting of mother coal lies 14 to 24 inches below the top of the bed, and a second parting generally appears 5 to 8 inches lower down. Above the upper parting the coal occurs in layers 3 to 6 inches thick, with partings of mother coal between them.

The immediate roof consists of a gray shale 15 to 110 feet thick. This shale does not stand well when the coal is removed, and the top coal is generally left as a roof, at least until the rooms are finished. The bottom is generally of clay, four inches to eight feet thick, below which is limestone. There is only one persistent band of impurity in the bed. This, which is known as the blue band, generally consists of bone or shaly coal and is found uniformly at a height of 18 to 30 inches from the bottom. Its thickness varies from $\frac{1}{2}$ -inch to $2\frac{1}{2}$ inches.†

The large number of squeezes which have occurred in mines of District VI would seem to indicate the presence of one or more thick beds of strong rock among the overlying strata. A study of the logs of numerous wells does not, however, show the presence of any continuous strong bed which would be a serious obstacle to the introduction of methods allowing a larger percentage of extraction. The State Geological Survey makes the following statement concerning the overlying limestone: "Over a large part of the area within 25 feet of the coal is a limestone cap rock which in places rests upon the coal, except for the draw slate that lies between. Where the lime-

* Andros, S. O., "Coal Mining Practice in District VI," Ill. Coal Min. Invest., Bul. 8, p. 11, 1914.

† Shaw, E. W., and Savage, T. E., U. S. Geol. Sur., Folio No. 185.

stone cap rock is not present within 25 feet of the coal it may be entirely absent, or lie at a considerably greater distance above the coal, amounting in some places possibly to as much as 100 feet." * The limestone cap rock is of variable thickness up to about 11 feet, the average thickness being 4 to 5 feet.† In some places sandstones are found at various distances above the coal, but none of these seems to be close enough to the coal to affect the choice of a mining method. In other words, it seems that there is no layer of rock, sufficiently near the coal to require serious consideration, which cannot be broken by careful attention to the proper methods. An examination of bore hole records of the Connellsville district of Pennsylvania, where the percentage of coal extracted is very high, indicates that there is more difficulty in breaking the overlying layers of rock in that district than would be experienced in most cases in District VI of Illinois. At a few mines an unusually wide room pillar is left in the middle of a panel for the purpose of limiting the area affected by a squeeze.

According to Table 3, all mines in the district, except strippings, are worked by the room-and-pillar method or by the panel method. Where the latter is employed, frequently no attention is paid to panel pillars so that the advantage of this method in the stopping of squeezes is largely lost. Practice is not uniform in regard to the number of rooms, which may be as low as 14 or as high as 30, turned from a room entry. The description of mining practice in this district‡ given in Bulletin 8 of the Coöperative Investigations says, "The immediate roof overlying the coal falls in slabs after short exposure to the air and top coal is usually left to protect it, but the cap rock is a tough coherent shale which does not break easily. The first mines opened in the district had widths of rooms and pillars unsuitable for this tough cap rock. New mines as they were opened adopted the dimensions of the older mines and a great waste has resulted through the loss of pillar coal. It will never be possible in this district to draw any considerable portion of the pillars where rooms 20 to 29 feet wide are driven with narrow room

*Cady, Gilbert H., "Coal Resources of District VI," Ill. Coal Min. Invest., Coöperative Agreement, Bul. 15, p. 83, 1916.

† Ibid, p. 32.

‡ Andros, S. O., "Coal Mining Practice in District VI," Ill. Coal Min. Invest., Bul. 8, p. 12, 1914.

pillars. Fear of yardage charges has been an important factor in maintaining the present improper dimensions. . . . With present dimensions when rooms have been driven 200 to 300 feet there is a large area of unsupported cap rock. If an attempt is made to draw pillars under such conditions a squeeze is usually started which often rides over room and entry pillars and sometimes affects a large acreage. In one mine 85 acres were squeezed; in another, 80."

Early operations were carried on without regard to the possible production of squeezes. Pillars were gouged out or entirely removed whenever the demand for coal seemed to excuse this procedure; a natural consequence was the occurrence of squeezes. At one mine there have been five squeezes of which two involved about 80 acres each, one about 40, one about 20, and one possibly 10. The present plan for the future operation of this mine contemplates leaving barrier pillars 150 feet wide along the important entries, and removing this pillar coal later. It is believed that this plan will confine roof movement to the worked out areas and that the entry pillars and barriers can be extracted later. This plan is much the same as that shown by Fig. 32, page 102.

At another mine a large squeeze approached within 125 feet of the air shaft and caused a depression on the surface which necessitated the regrading of a considerable amount of track, including the track scales. Practice at this mine represents one extreme, since no attempt is made at room-pillar drawing beyond driving cross-cuts about 30 feet wide at the ends of rooms, and as much coal as possible is taken on the advance. An attempt will be made to take out barrier and entry pillars on the retreat. Rooms are driven 25 feet wide on 45-foot centers, and cross-cuts are 25 feet wide. The excavated area is about 50 per cent and the top coal, which is only 18 inches thick, is left up.

The figures for recovery of coal given in Table 3 are unquestionably too high although they were based on the best information available at the time. The average percentage of extraction in District VI is not more than 50 per cent, and it is probably nearer 45 per cent. The maps of mines may show an excavated area of 50 per cent or even more, but they do not take into account the unmined top coal. The thickness of coal taken out is generally about 7 feet and top coal ranging from a few inches to 4 or 5 feet in thickness is left. Even if the top coal is ignored, the extraction

is not so high as the estimates generally indicate because of losses in squeezed areas and boundary barriers.

Special investigations on the subject of recovery made at several mines in Franklin County gave results which are summarized as follows:

At one mine, the recovery in worked out areas where pillars are not drawn is about 65 per cent; where the pillars are taken, it is about 75 per cent.

At another mine, close observations were made in connection with a study of subsidence. In a panel where the extraction was considered good and possibly above the average for the mine 40 per cent of the coal is left as pillars. Two feet or 20 per cent of the thickness of the bed is also left as top coal, and the loss from this cause would be 20 per cent of the remaining 60 per cent or 12 per cent of the total. The total loss is then at least 52 per cent. No attempt has been made to extract room pillars, but some entry pillars are taken, and top coal is taken over the area in which these pillars are drawn.

At one of the mines where the thickness of the coal is greater than the average, little pillar work has been done. The coal varies from $9\frac{1}{2}$ to nearly 16 feet in thickness,* and about 9 feet of it is taken out. Generally about one foot of coal is left on the bottom to avoid the possibility of taking up a bed of "black jack" which is not easily distinguishable from coal. This black jack is probably a coal of very high ash content. The leaving of bottom coal results in the elevation of the working place in the bed so that the top coal left is only 3 feet or even less in thickness. In a few places in this mine some pillar coal has been taken out. Where this was done, break-throughs about 24 feet wide were driven at the ends of the rooms. Then work was commenced at the ends of the pillars, and coal was taken out by pick work. This work seems to have been successful, but it has not been followed systematically. The largest number of pillars which have been taken together was six, and no attempt was made to obtain a break in the roof.

*Cady, G. H., "Coal Resources of District VI," Ill. Coal Min. Invest., Bul. 15, p. 58, 1916.

The leaving of coal on the bottom is a practice followed at only a few other mines. In some cases where the blue band is thick the mining is done above it, and a portion of the upper part of the bed, ordinarily included in the top coal, is taken down. At two mines where this method is followed in part, the blue band and the coal below it are left in where the blue band is thick and the top coal is taken to within about ten inches of the roof, at which point there is a parting. Greater care is required to prevent the breaking of the top coal where this is done.

At one of the mines in the southern part of Franklin County a little pillar coal is drawn, though pillar coal is not depended upon for an important part of the output. Rooms are 25 feet wide with 20-foot pillars. Rooms are holed through into those of adjoining panels. When the rooms have reached their full lengths, cross-cuts 24 feet wide are driven across the ends of the pillars. In addition to these cross-cuts the pillars are probably slabbed to some extent. The coal is about 9 feet thick, and about $1\frac{1}{2}$ feet of top coal is left up. No bottom coal is left. The barrier pillars are about 100 feet thick. Break-throughs are 21 feet wide. It seems hardly proper to speak of this kind of work as the extraction of pillar coal, but it represents a practice which is common in this district.

At one of the mines, pillars are drawn, beginning in the middle of a panel, in six rooms at a time; then another group of six pillars is attacked, one pillar being left untouched between the groups. This is simply another method of attempting to get as much coal as possible before being driven out by a fall of the top, and is not an attempt to break the cap rock.

Systematic work in the recovery of pillar coal as done at one of the mines is illustrated in Fig. 4. The mine is operated on the block system commonly called the panel system, though the panels are not kept sufficiently isolated to warrant the use of the term. Cross entries are driven at intervals of 1,370 feet, and panel entries are driven through from cross entry to cross entry. On each pair of panel entries twenty-eight rooms are turned to each side on 40-foot centers. A barrier pillar 125 feet wide is left along the cross entry. After the room entry is driven the rooms are necked, but only the first

fourteen rooms on each side of the room entry are worked, and these are finished before the rooms at the other end of the panel are driven. Break-throughs are normally 11 feet wide; those at the ends of the rooms are 24 feet wide. Pillar drawing is commenced when the rooms at one end of the panel are finished, and the coal is taken out through the

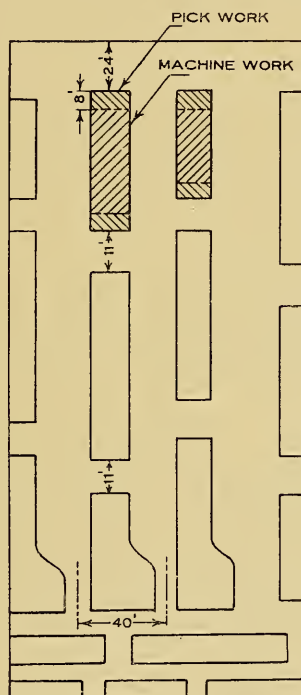


FIG. 4. PILLAR DRAWING IN FRANKLIN COUNTY, ILLINOIS

first cross entry, that is, the one next to room No. 1. The coal from the remaining portion of the panel is taken out through the next cross entry, that is, the one next to room No. 28. The advantage of this method lies in the fact that the extraction of the coal from the second half of the panel is not interfered with, as far as haulage and ventilation are concerned, by movements caused by pillar drawing in the first half. The pillar coal is attacked first by the driving of 24-foot cross-cuts at the ends of the rooms; then other cuts are made in the pillar

with the breast machine in such manner as to leave stumps between the cuts and the break-throughs. These stumps are removed as far as possible by pick work, but the miners are not always able to finish as much work with the machines as is desired since this work is frequently interrupted by movements of the roof. As much of the coal as is possible is then taken out with picks. While the work at this mine is as systematic as that at any mine in southern Illinois, the company has no exact record of the amount of pillar coal extracted, but it is known that the pick-mined coal amounts to approximately 10 per cent of the output. When the pillars are not drawn, the recovery is estimated by the operators to be about 65 per cent; when they are taken, the estimates run about 75 per cent.

Through the courtesy of the Franklin County Coal Operators' Association (Illinois), data have been made available regarding the extraction of coal in that county as presented in the following paragraphs:

The coal mined is the No. 6 bed of the State Geological Survey classification. Measurement of 113 sections taken in twelve of the largest mines in the county gave an average thickness of 9.2 feet of coal, the average minimum thickness for the same twelve mines being 8 feet, and the average maximum thickness 10.64 feet. The blue band, which is characteristic of the No. 6 coal bed, varied from $\frac{1}{4}$ to 2 inches in thickness, and its average distance from the floor was 21.5 inches. Owing to the difficulty of keeping up the shaly material above the coal bed, the top coal is almost universally left as roof protection, and, up to the present time, very little of this top coal has been recovered, although some operators are expecting to recover it at a later date in connection with pillar drawing. In one of the twelve mines from which the data were obtained, top coal was not left in the rooms. This, however, is exceptional practice, the average thickness of the top coal left in the twelve mines being $1\frac{1}{2}$ feet. The average thickness of coal mined was 7.46 feet, and the average tonnage per acre to January 1, 1916 was 6,627 tons. This is equivalent to 40.7 per cent extraction, if it is assumed that all the 9.2-foot bed is available for ship-

ment, or to 41.6 per cent if it is assumed that the blue band and refuse discarded in the loading, or 0.2 foot, is deducted from the thickness of the bed. A very careful estimate for each of the twelve mines noted, made by dividing the total amount of coal in the area mined up to January 1, 1916 into the actual shipments since the mine began operating, gave percentages of extraction varying from 37.7 to 49.5, or an average of 41.4 per cent.*

For six of the twelve mines, data were available for the average percentage of extraction in the portion of the bed actually mined; that is, the total thickness less the top coal left up to protect the roof. This average is 48.65. These mines are all comparatively new mines, and in only a few cases has any portion of the workings reached the boundary so as to permit drawing the pillars in return workings. At many of the mines it is hoped to increase the percentages of extraction through subsequent pillar drawings, but the amount of such increase is, of course, problematical. In many instances squeezes have already occurred, but as a general thing only the room pillars have been affected.

The twelve mines under discussion are representative of the practice in Franklin County and to a great extent of that of southern Illinois. In a number of these mines experiments are now being conducted to determine in what respect present methods of working may be modified to yield a larger percentage of extraction. Although these mines are operating under practically the same physical conditions and all on the panel system, the variation in the detailed operations, such as the number of rooms per panel, or the width of barrier pillars, indicate the necessity for a critical comparative study of details to determine the best method for the given conditions.

Investigations in Williamson County supplied the following facts:

At one mine rooms are 21 feet wide on 40-foot centers. From 2 to 2½ feet of top coal is generally left up on the advance. In one part of the mine the coal is 11 feet thick and only 7½ feet of it is taken out. It is estimated that 40 to 50 per cent of the pillar coal is won. The top coal, which is the

* The Peabody Coal Company reports that the percentages of extraction at its four mines in this district are 67, 63, 55, and 55.

best part of the bed, is taken out when the pillars are drawn. There are no definite records on recovery, but it is probable that 55 or 60 per cent is gained. If $7\frac{1}{2}$ of the 11 feet are removed, and 40 per cent of the pillars and all the top coal over the area in which the pillars are drawn are removed, the extraction is about 60 per cent.

In another mine a rather high percentage of extraction is attained because of favorable conditions which permit the leaving of small pillars. The coal is 9 feet thick and the depth only 100 feet. Rooms are 24 feet wide on 35-foot centers. In some cases top coal, about 20 inches thick, is left if the machine runners think the top is insecure. Probably from 65 to 70 per cent of the coal is taken out. No pillar work could be done with rooms and pillars of these dimensions, but on one side of the mine 15-foot pillars are now being left with the intention of taking them out on the retreat.

At some mines in the western part of Williamson County considerable trouble has been experienced, because large quantities of water enter when the top is broken. The cover here is only about 100 feet thick and there are only 3 to 4 feet of solid rock. The rooms of one mine are 20 feet wide and are driven on 40-foot centers, although they are sometimes crowded. Some rooms were driven on 32-foot centers, but the pillars were not sufficient to prevent squeezes. Entries are 12 feet wide and entry pillars 20 and 25 feet wide. The coal is 5 to 11 feet thick with an average thickness of 8 feet. Top coal averages 20 inches in thickness. Above the bed is a shale; above this is a so-called soapstone, ranging from 2 to 8 feet in thickness and averaging about 4 feet; and above this is a black shale. In some places a draw slate from 1 to 2 feet thick occurs above the coal, and above this is limestone 1 to $3\frac{1}{2}$ feet thick. Where is no draw slate, there is no limestone. An unconsolidated sand is found in some places above the coal. The pillars are sometimes slabbed a little to compensate for the coal left in entry and barrier pillars. When this slabbing is done, the extraction amounts to about 50 per cent. In some cases extra cross-cuts are taken, and the extraction is thereby increased to about 75 per cent. On the whole, the extraction is estimated to be about 60 per

cent,—an estimate which is probably reliable since the work is more carefully done here than at many other mines of the district. Where more than 60 per cent of the coal is taken out in this mine, the roof breaks and water enters in large quantities. Although the amount of water is influenced by precipitation, the flow is continuous. In one case where the rock had been broken and a large quantity of water had entered the mine, it was thought that the strata was drained to some extent and that it would be possible to allow the top to break at a slightly higher elevation. It was found, however, that the new break allowed a large amount of water to enter the mine, and it has been impossible for the company to do any pillar work. It is planned, as some of the workings reach the boundaries, to draw pillar coal. In these cases pumps will already have been installed, and it will be possible to conduct the water to these. The water, moreover, will be entering in abandoned places and not between the workers and the shafts. Some of the workings are now not far from the boundaries, and the plan can be put into operation in the near future.

At another mine the coal is 9 feet thick, the top is of white shale, and the bottom of fire clay. The top coal is about 2 feet thick. The mine is operated on the panel system. No pillars are drawn until the rooms on an entry have been finished; then a cross-cut three machine-cuts wide, or about 20 feet, is driven at the ends of the pillars in about half the rooms on the inside end of the stub. Following this, another cross-cut is made farther back in the pillar leaving a stump about 10 feet wide. The distance between the first and second cuts varies according to conditions, or according to the judgment or inclination of the machine men. If it is made farther back, some of the pillar coal is lost. This operation is repeated until the first break-through in each pillar is reached, the remainder of the pillar being left standing until all the rooms on the stub are finished; then the room stump and the entry pillars are drawn. No effort is made to obtain a break in the roof, and the leaving of stumps of pillars is likely, by partially sustaining the roof, to bring on a squeeze. At present the driving of rooms without necks is being tried, the purpose being to avoid payment for narrow work; and it is believed there will not

be sufficient difference between the support left under that system and that left under the present system to endanger the entries. These rooms without necks are turned six machine-cuts wide and are widened to seven cuts beyond the first cross-cut.

At another mine the average thickness of the coal is 9 feet, 4 inches. The top coal is about 2 feet thick and is left up until the pillars have been partly drawn back, being taken down just before the track is removed. There is generally a good parting between the main bed and the top coal, which is said to be poorer than the main bed. Above the bed is shale of unknown thickness, which has never broken high enough to expose any other rock above it. This shale slacks when exposed to the air. It does not form a very good top and most of the entries are timbered. The bottom is generally of clay, but in some places limestone appears next to the coal. Rooms are 20 feet wide on 30- to 35-foot centers and are 185 to 190 feet long. Stub entries are turned on 400-foot centers, and 16 to 18 rooms are turned from a stub. The room pillars are gouged to a considerable extent. It is planned that all the rooms in a panel shall be driven to their full length before pillar drawing is commenced, but this plan is not always followed, and squeezing sometimes commences before all the pillars can be attacked. This, of course, is promoted by the gouging of room pillars. When the rooms are finished, cross-cuts 20 feet wide are driven at the face with breast machines. The rest of the pillar work is generally done with picks though machines are used when possible. Movement of the roof, however, generally interferes with machine work after the first cross-cut. The pick work generally consists of slabbing along the sides of pillars, but machines are sometimes used. Squeezes have always been confined to the panels, and no entries have been lost until the entry pillars have been drawn. A careful computation, based upon a comparison between the actual area worked and the number of tons hoisted, shows that the extraction at this mine has been 48.89 per cent. This is one of the most thoroughly worked mines in the No. 6 bed, and the estimation of percentage of extraction is undoubtedly as close as any that has been made. The results found furnish one of the reasons for the

statement that extraction in most mines of the district is less than the operators of the mines believe it to be.

At a Perry County mine the general system is the same as in Williamson County. Room entries are driven through from cross entry to cross entry. A somewhat closer adherence to the panel system is to be noted, however, in that 25-foot pillars are left at the ends of rooms. Rooms are 24 feet wide on 60-foot centers, and they are driven 250 feet long. Break-throughs are staggered. When the room is completed, an 18-foot cross-cut is driven through the pillar at the end. Top coal, about 3 feet thick (the best of the bed in quality* as it is at the mine last mentioned), is left up until pillar drawing commences. Pillar drawing is commenced in the middle of the panel. After the completion of the cross-cut at the end of the first pillar attacked, the top coal is loosened by a light shot near each rib. Work on the pillars is then prosecuted by making a cut through the pillar, if the condition of the top will permit, wide enough to leave an 8-foot stump at the end of the pillar and another of the same dimension next to the nearest break-through. These two 8-foot stumps and whatever is left by the machines are taken out by hand work. Two men are used on solid work and two on the machine. This method has been found successful and a considerable amount of pillar coal has been recovered, but pillar drawing is not a necessary part of the system and is not always carried out. Where the pillar coal and the top coal are taken, the recovery is said to be from 75 to 80 per cent of the coal in the area actually worked.

Several plans are now being tried or considered for the more nearly complete extraction of the coal in this district. One of these, which has so far been given only an incomplete test, is a panel long-wall method. Double entries, which would have been the room entries of a panel under the ordinary methods of operation, were driven 340 feet long. At the end, two rooms were driven on each side separated by 25-foot pillars. The rooms at the extreme end of the block on each side were 9 feet wide and the ones further back were 18 feet wide; each was 200 feet long. They were connected at the ends so that ventilation was obtained, the course of the air current being as shown

*For a discussion of the differences between the top coal and the remainder of the bed see "Chemical Study of Illinois Coals," by S. W. Parr, Ill. Coal Min. Invest., Bul. 3, p. 49, 1916.

in Fig. 5. Then the outby ribs of the 18-foot rooms were worked as long-wall faces by continuous-cutting chain machines making a 6-foot cut. The top behind the working face was propped. It was the intention to support the immediate roof until the face had advanced some distance and then to make an attempt to break the overlying rock by the withdrawal of the props. This plan was found to be impossible,

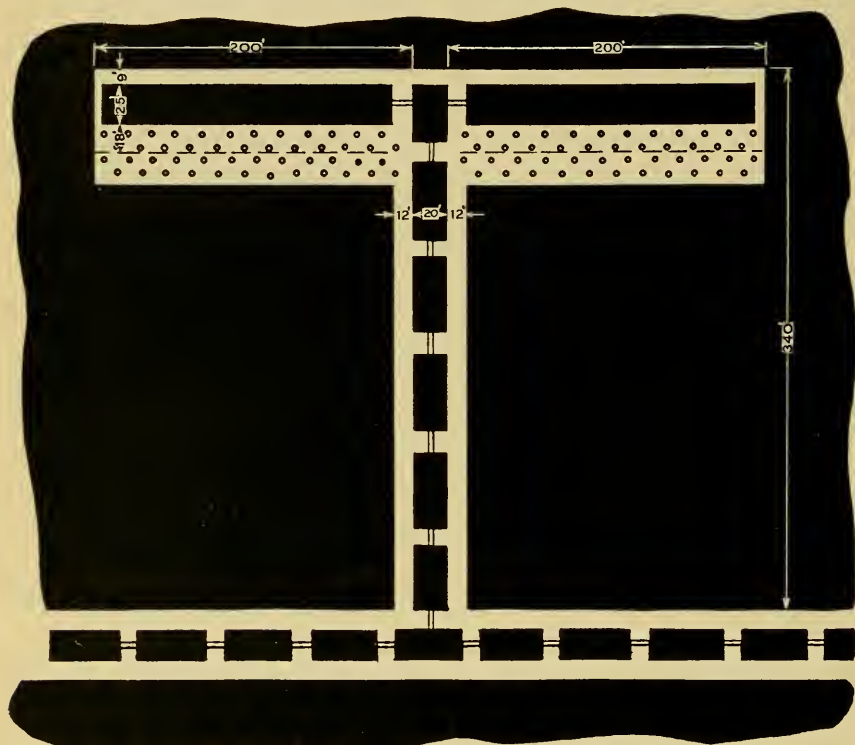


FIG. 5. PANEL LONG-WALL

however, as the top fell when the face had advanced only about 40 feet. Other conditions made it necessary to discontinue the experiment temporarily. In operating by this method, sprags were placed in the cutting behind the machine to prevent the premature fall of the coal. No trouble was experienced in getting the coal down; it was produced very rapidly and was easily handled. At present it is not known whether the top can be broken along the desired line, but it will be seen that this line is only 400 feet long and that it is interrupted in the

center by the entry pillar. Even if it is not possible to break the top and to work the coal back continuously on two longwall faces, it seems that the attack can be repeated farther back in the block and that coal can be produced as cheaply as by the ordinary method; also that a much higher percentage of extraction can be attained. If it should be necessary to follow the method by repeated attacks on the block, there would be some resemblance to the "single room" method successfully worked in West Virginia.

Various other plans for higher extraction have been suggested and some have been partly applied, but the great demand for coal, which has been stimulated by the European War, has caused coal producers to concentrate all their attention upon the immediate production of a large tonnage. Anything in the nature of experimental work will be postponed until the return of more nearly normal conditions, but there is reason to believe that successful efforts will be made to increase the percentage of extraction and that the present large loss will be greatly decreased.

16. *District VII.*—The coal worked in District VII is the No. 6 bed on the west side of the Duquoin anticline. The thickness varies from $2\frac{1}{2}$ feet to 14 feet and averages about 7 feet. There is a well defined parting plane in the coal about 18 inches from the roof. Where the roof is of black shale and where the coal is 7 feet or more in thickness, the upper bench or "top coal" is left. The roof is a non-calcareous black shale, a calcareous gray shale called locally "white top" or "soapstone," an unconsolidated dark gray or black shale called "clod" and made up of fragments of varying size and hardness extremely difficult to support, or a hard gray limestone called "rock top." A poorly defined cleat or cleavage in the coal may be seen in some places. The floor throughout the district is of fire clay which generally heaves when wet.

The thickness of the coal is almost ideal for easy working and for large production; some of the mines have obtained daily capacities which rank among the highest in the world. The older mines have been worked without much regard to system, but the newer ones are more carefully planned. The planning, however, is directed toward large daily production rather than toward a high percentage of extraction.

Varying roof conditions often make different entry and room

widths necessary in different sections of a mine. In many mines the entries and rooms under rock top are too wide and the pillars too narrow,—a condition responsible for squeezes which sometimes have endangered even the shaft. Squeezes have occurred in thirteen of the twenty-five mines examined in this district; they have generally begun in sections in which the roof was of limestone. In mines in which the rooms are not frequently surveyed there is no definite knowledge of room-pillar width except at cross-cuts. Table 3 gives dimensions of workings at each of the mines examined.

In ten of the mines examined where the immediate roof was of thick black shale, top coal was left to prevent variations of temperature and humidity from affecting the shale of the roof proper, which spalls badly when exposed to the air. Where no top coal is left, this black shale usually falls with the coal or is drawn. Where there is less than four inches of shale between the coal and the limestone, the shale is drawn. In some mines where the latter is more than four inches thick it is propped; in others it is drawn, unless it is more than two feet in thickness.* The Peabody Coal Company reports extractions at its mines in this district of 65, 62, 60 and 50 per cent. At the twenty-five mines examined, the average estimated percentage of recovery furnished by the operators was 55.5 per cent. So far as is known, no efforts have been made to extract a higher percentage of coal. The reason for this attitude is to be found partly in the condition of the surface, which in many places is so nearly level that any noticeable subsidence disturbs the drainage and effects the value of the surface for agricultural purposes.

At present the abandonment of a large percentage of the coal is a result of the difficulties experienced in attempting to secure satisfactory agreements with the owners of the surface. In some cases the owners of coal rights are not the owners of the surface and are not free from responsibility for surface damage. Since the operators have found that the estimates of damage caused by subsidence are likely to be very high, it has become the custom to operate the mines under methods which will avoid subsidence. Unfortunately it has not been possible to estimate the exact amount of coal which must be left in the ground, and squeezes and subsidences have sometimes occurred when it was thought that sufficient coal had been left.

* Andros, S. O., "Coal Mining Practice in District VII." Ill. Coal. Min. Invest., Bul. 4, p. 11, 1914.

Even where low value of the land or good drainage reduces the cost of possible injury to the surface by subsidence, no effort is made to secure a higher extraction. The occurrence of squeezes is feared, and experience shows that the only way to prevent them without radically changing the system of mining is to leave large amounts of coal in the form of pillars. One company which was formerly getting 50 to 60

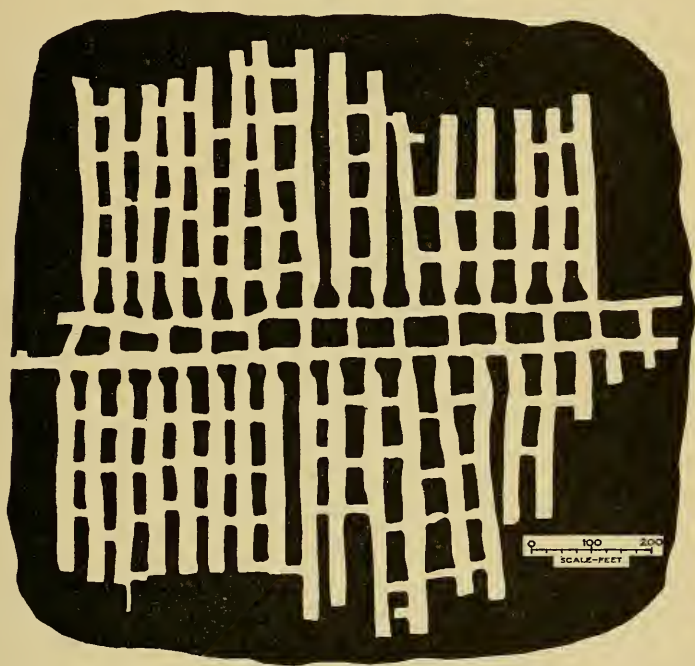


FIG. 6. PLAN OF AN OPERATION IN MACOUPIN COUNTY, ILLINOIS, SHOWING EXTRACTION IN A LIMITED AREA

per cent of the coal with frequent squeezes and subsidence has changed the dimensions of rooms and pillars so that now only 40 to 50 per cent is obtained. Thus far, with the new dimensions, squeezes have not occurred. No effort has been made to extract pillars systematically with the purpose of breaking the cap rock and thus preventing a squeeze by relieving the stress on the pillars, but there is nothing to indicate that this plan could not be carried out.

The plan of a portion of one of the mines is shown as Fig. 6. This

may be taken as a fairly typical projection of large mines in this district. In many of the mines, including parts of the one illustrated, the workings are on a panel system, but probably not enough attention is paid to the matter of leaving pillars sufficiently wide to prevent the spread of squeezes beyond the boundaries of the panels. This illustration is presented, because the mine was surveyed with unusual care in connection with an investigation of subsidence which is being carried on by the Coöperative Coal Mining Investigations. The rooms and pillars are about 30 feet wide; this dimension was adopted with the belief that the top would be held up by pillars of this width left between 30-foot rooms. It had been found that the roof would fall if 25-foot pillars were left between 25-foot rooms. In the restricted area measured, the extraction amounts to 59.2 per cent of the area worked; that is, 40.8 per cent of the area has been left as pillars.

17. *District VIII.*—In District VIII, seams 6 and 7 are mined. In both seams there are numerous rolls of roof and floor called “faults,” or “horsebacks.” In many cases the roll completely displaces the coal.

Seam 6 averages 6 feet in thickness. Near Danville the immediate roof is of grayish black shale about 6 feet thick. This shale, lying between the coal and a cap rock of dark gray nodular limestone, makes a roof which is easy to support. In the vicinity of Westville and Georgetown, the immediate roof is generally of gray shale which shows no distinct bedding, has little cohesion, falls in conchoidal masses, and is extremely difficult to support. Stringers of coal, furthermore, extend from the seam proper into the roof material and render the task of supporting the roof more difficult. Occasionally there are 3 to 4 inches of black shale between the coal and the gray shale which forms the cap rock. Wherever this black shale is broken, air and moisture disintegrate the gray shale cap rock, and the roof becomes unsupportable. In all parts of the Danville district the floor is of soft fire clay.

Seam 7 varies in thickness from $2\frac{1}{2}$ to $5\frac{1}{2}$ feet, the average being 5 feet. The coal has two benches separated by a clay band one inch thick, which persists throughout the bed from 6 to 8 inches above the floor. This bed also has numerous rolls.

While the stripping operations, which are important in this district, are conducted in the No. 7 bed, the largest underground oper-

ations are in the No. 6 bed. The mines are operated on the room-and-pillar method, or a modification of it, but the numerous rolls in the roof prevent close adherence to the system. The frequent occurrence

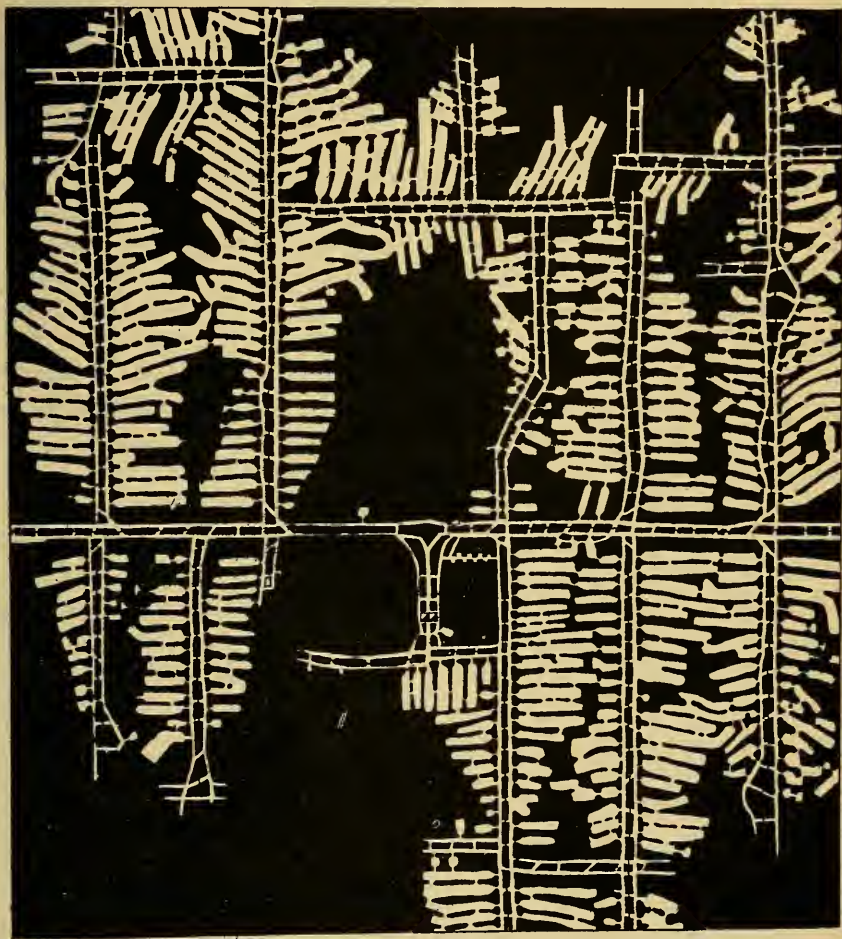


FIG. 7. PLAN OF MINE IN VERMILION COUNTY

of rolls has a marked effect upon the manner of driving rooms. In a roll area it is difficult to support the roof, and the expense of driving through the hard rock of the roll is great; consequently, when a roll is encountered in driving rooms it is customary to change the direction

of the room and to drive it parallel with the roll until the coal resumes its normal condition, as shown in Fig. 7, which is a map of a mine typical of the district. Often it is necessary to abandon a room before it has been driven to its proper length. Since the rolls are of frequent occurrence, the amount of coal that may be gained in any section of the mine is problematical; consequently, the operator, on reaching that portion of the coal where the seam regains its normal thickness, will attempt to get as much of the coal as possible during the first working. Little attempt is made to preserve a constant room-pillar width, and the practice of gouging pillars is common in the smaller mines.* No systematic pillar drawing is attempted, because with present practice there is little pillar coal left to draw when the rooms are driven to their full length. The roof is so treacherous, especially in the vicinity of the rolls, that it is not safe to leave wide spans of roof unsupported by pillars.

The width of room pillars at the mines examined varied from 4 to 16 feet, and room widths varied from 21 to 43 feet. Table 3 gives dimensions of workings at each mine examined. Very narrow room pillars were found in mine No. 91, where the following dimensions were recorded; room centers, 47 feet; room widths, 43 feet; room pillar width, 4 feet.

Although pillar gouging in the district has resulted in a high percentage of extraction from the bed in the first working, it has caused a subsequent loss of coal through squeezes due to narrow pillars. The average extraction for the six mines examined, as reported by the operators, is 70 per cent. Table 3 gives also the percentage of the bed extracted at each mine. These percentages were calculated from measurements made in the mines and were checked by records of production per acre obtained from the books of each operating company and by planimeter measurements of mine maps. The Peabody Coal Company reports an extraction of 66 per cent at its mine in this district.

At one of the mines, almost all the pillar coal was extracted after all the advance work had been done, and the roof was supported largely by the rolls which occurred at intervals of 60 to 100 feet. At another mine pillars are being extracted, and it is estimated, that the total recovery will amount to about 85 per cent.

* Andros, S. O., "Coal Mining Practice in District VIII," Ill. Coal Min. Invest., Bul. 2, p. 16, 1914.

18. *Conclusion.*—It will be seen that nearly all the work in Illinois described as pillar drawing is unsystematic. It is merely incidental to the mining of room coal, and preparation for it is rarely made in laying out the mines. There are no apparent reasons, so far as physical conditions are concerned, except in a few instances, why plans could not be made for leaving pillars large enough to support the top during the advance work and for recovering the pillars on the retreat. Squeezes could thus be avoided, and the percentage of extraction could be increased materially. The commercial conditions which seem to make such a course difficult could probably be overcome, except in those cases in which subsidence of the surface subjects the operators to claims for damages in excess of amounts which would seem to be reasonable compensation for the injury done. The law covering payments for damages due to subsidence ought to be made so clear that there could be no doubt concerning the amount to be paid, and this amount should be limited to a fair compensation for the injury actually done.

At present there is promise of a considerable improvement with regard to the percentage of coal extracted from Illinois mines. The subject is receiving more and more attention on the part of coal producers and careful planning of the work with a view to high extraction as well as to low cost will follow as the natural result of greater interest on the part of the operators.

CHAPTER III

METHODS AND RECOVERY IN THE UNITED STATES

19. *Early Methods in the United States.*—This chapter presents a discussion of early methods of mining coal in the United States, information regarding the percentage of coal recovered in different districts, and descriptions of the most advanced methods employed for obtaining high extraction, especially those which are applicable to conditions in Illinois. In collecting this material all available sources of information have been utilized. The descriptions of methods have been taken largely from the technical literature of coal mining, but in instances in which the correctness of the description seemed in doubt, or in which statements concerning the percentage of extraction seemed to need verification, the subjects have been reviewed by persons familiar with the local conditions.

In response to the large number of inquiries sent out, many persons have furnished the desired information in as nearly complete form as possible, but in many cases there has been available no authentic information on the subject of recovery. The estimates are necessarily more or less approximate because the conditions are such that it is practically impossible to obtain correct values, or the subject has not been considered of sufficient importance by the operators to warrant the expenditure of the time and money necessary for obtaining the values. It is believed that the values for percentage of extraction given in the following pages represent the most reliable information obtainable on the subject, but they are not presented as being absolutely correct.

At the time mining was begun here, this country was a colonial possession of Great Britain; the methods of mining to which immigrants were accustomed were those of Great Britain, and the application of these methods to mining problems in America was a matter of course. The development of the early English methods is discussed in the appendix. The coal miners of this country, furthermore, have been for the most part men who received their training in the work in England, Scotland, or Wales, or children of such men, and not until

a comparatively recent date did these miners lose their dominance in the American coal fields. The conditions under which coal was found in this country were also not very different from those in Great Britain. It was natural, therefore, that bituminous coal mining practice in this country should correspond to that of Great Britain at the time the industry began here.

Mining in this country was begun in the Richmond (Virginia) basin about the middle of the eighteenth century. There seems to be no clear record of the methods followed, but it is known that a pillar system was employed, and that, as the coal was reached in some places at a depth of several hundred feet, a considerable amount of the coal was left in the ground. It is said that the pillars were to be extracted on the retreat, but no definite record is found to indicate that this was done.

Western Pennsylvania was the next district to take up coal mining on an important scale. Maryland and West Virginia followed, basing their early methods for the most part on what had been done in Pennsylvania.

20. *Pennsylvania*.—The early history of coal mining in the western Pennsylvania district is typical of that in other sections of the country, having a similar hilly topography. When coal mining was commenced, an abundant supply of coal was found outcropping on the hills in the neighborhood of Pittsburgh, and these seams were attacked by numerous small mines on the outcrop. As the workings were extended under cover, the single entry system was followed, and as it was impossible to obtain good ventilation with this system, the rooms were driven to only a short distance, and the entry itself was not long. Later the double entry method was employed, in which two parallel entries were used, respectively, for intake and return air. Since the distance to which rooms could be driven was limited, it was impossible to work any large territory by this method; hence, as the size of the mines increased, the cross entry system was introduced. The underground developments were the same whether the coal was reached by drifts, by slopes, or by shafts.

Among the many experiments tried in the Pittsburgh bed was that involving the use of double rooms with double necks, or of double rooms with single necks, but the amount of timber required for posts made these methods too expensive and by 1906 they were in use in a

very few mines. The long-wall system also was tried and abandoned.*

No record has been found of the time at which the drawing of pillars was commenced, and it is probable that this method was followed more or less from the beginning in such mines as were systematically developed.

Toward the end of the last century, the double entry system had been further developed by the turning of room or butt entries from the cross or face entries. In some mines a few of the entries were driven to the boundary, and then all the rooms were opened at once, but some of the center rooms would sometimes reach their limits before

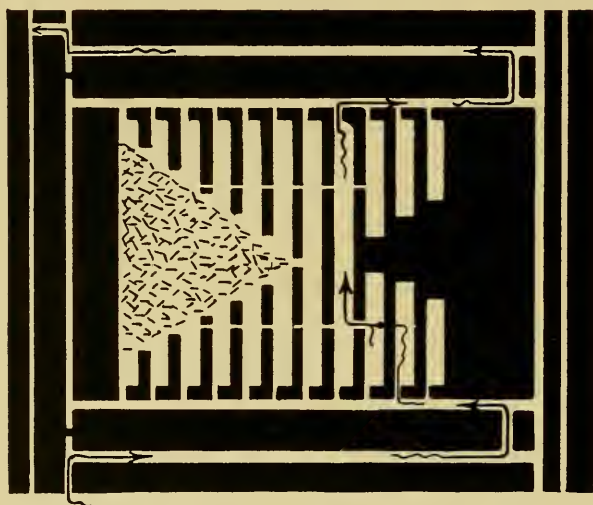


FIG. 8. OLD METHOD OF ROOM-AND-PILLAR IN PITTSBURGH, PA., DISTRICT

those whose pillars should have been drawn first. In other cases only the inby half of the rooms on each entry was turned first, while in still others one entry was completely exhausted before any side work was done on its parallel entry. In most cases, a room-and-pillar method was used with double entries, each about 9 feet wide. Main entries were separated by a pillar 51 feet wide with cut-throughs for ventilation. The main entries were driven on the butt of the coal, and face entries were turned from them about 1,000 feet apart. From these face entries, secondary butt entries or room entries were driven

*Dixon, Charlton, "A New Method of Coal Mining," *Mines and Minerals*, Vol. 27, p. 32, 1906.

about 400 feet apart. Rooms about 20 feet wide and 200 feet long were turned on the face of the coal. The room necks were 21 feet long and 9 feet wide. Room pillars were 15 or 20 feet wide, according to the cover above the coal. The rooms were turned from the butt entries as fast as these were driven, room pillars being drawn as mining pro-

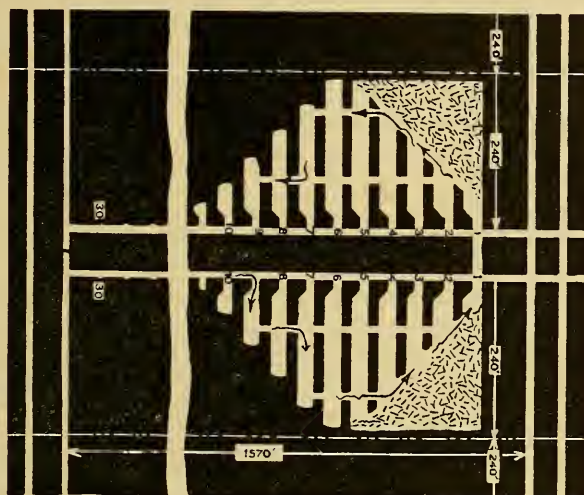


FIG. 9. IMPROVED METHOD OF ROOM-AND-PILLAR IN PITTSBURGH, PA., DISTRICT

gressed.* The objectionable features of this method are:—poor ventilation, dangerous gob, entries filled with fallen dirt requiring expense for cleaning up, maximum extent of track for the minimum quantity of coal, thus greatly increasing the cost of animal haulage, loss of thousands of tons of coal, compulsory driving of narrow work in room turning, and squeezes, which damage the coal and greatly increase the hazard of mining.

The difficulty of ventilation becomes most serious when the rooms from one entry are holed through into those approaching from a neighboring entry (see Fig. 8). Often this will occur two-thirds of the distance up each of these entries; thus all the pillars below the short circuit are deprived of proper ventilation at a point where it is constantly needed. After the pillars are drawn and the roof falls, there is no appreciable movement of air through the gob, and it often fills with explosive gas.

*Auchmuty, H. L., in *Coal and Metal Miners' Pocket Book*, 9th ed., p. 295.

A method described by Dixon, in the article previously referred to, was soon adopted with various modifications, although it is possible that it was already in use in one or more places at that time. According to this method the territory was laid off into blocks (see Fig. 9) 1,570 feet long, allowing for a barrier pillar 200 feet wide along the main entries and for another 200 feet along the next pair of face entries. A pair of room entries separated by a 54-foot pillar was driven through the center of this block, and thirty rooms were turned from each entry. Rooms were 240 feet long, about 26 feet wide, and were driven on 39-foot centers, thus leaving 13-foot pillars. Room turning was begun at the inby ends of the room entries, a reversal of the common practice of the time. The drawing of pillars was commenced as soon as the rooms were finished, and the line of break was kept at the proper angle by carefully timing the extraction of pillars. In this method the ventilation was considerably better than in the earlier method; but the air current, after passing through the district of pillar work on one room entry, went through the advancing rooms turned from the other. This difficulty was avoided in later methods by exhausting one room entry before room work was done on the other. The roof in the entries was easily maintained because the entries, with the exception of those on which rooms were being worked, were in solid coal. At the finishing of a block the minimum of track was in use for the minimum of coal passing over it. Track was not left in place awaiting the withdrawal of entry pillars; therefore it was not exposed to the corrosive action of mine water. Since the room pillars were attacked immediately, there was little danger of deterioration of coal or of trouble from falls. Most of the props could be recovered as they had not been subjected to any great pressure. Under the old system 50 per cent of the wood rails in rooms were lost while awaiting the attack on the ribs, and about 75 per cent of the posts were lost.

Referring to the conditions and the methods employed in mining, F. W. Cunningham* said in 1910:—"The operator in the Pittsburgh coal field, with the price of coal where it is to-day, must get the largest percentage of lump with the least amount of fine coal, and this by machine mining, in order that he may compete with coal operators in other fields." The rooms, therefore, are made as wide as possible to obtain the greatest percentage of lump coal, and the pillars are left

* Cunningham, F. W., "The Best Methods of Removing Coal Pillars," Proc. Coal Min. Inst. Amer., p. 275, 1910; p. 35, 1911.

as narrow as possible, because the greatest percentage of crushed coal comes from them. This fact explains why the use of narrow rooms and wide pillars, common in the Connellsville district, does not appeal to operators in the Pittsburgh district. It also explains the loss of much pillar coal, because a period of dull market results in the stopping of pillar work and only large coal from the rooms is marketed. A large number of rooms, accordingly, may be driven up to their limits

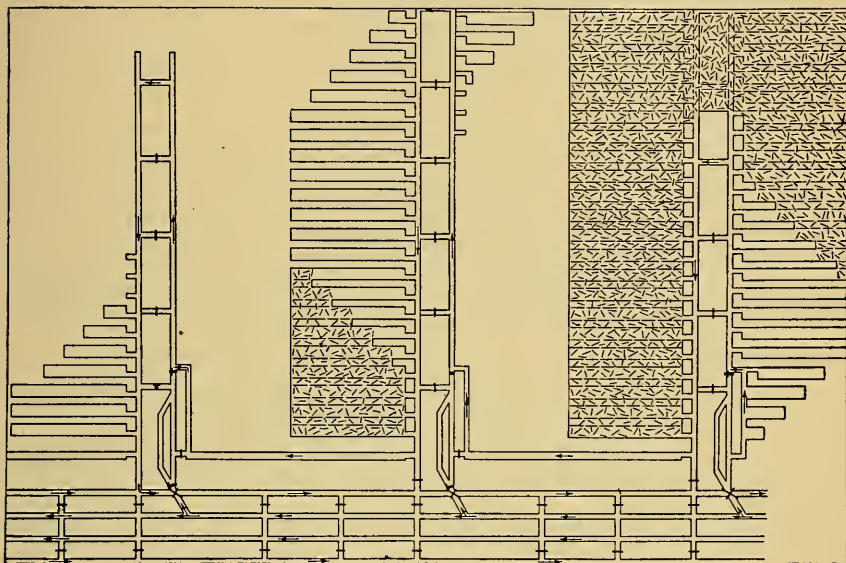


FIG. 10. MODERN METHOD IN PITTSBURGH DISTRICT

without the extraction of room pillars, and the recovery of these pillars is unprofitable after the rooms have stood for a number of years.

Fig. 10 illustrates the series of operations incident to one method of extraction of stump and chain pillars. In this method, rooms are turned and worked out progressively along one of a pair of room entries, probably the last, the pillars being drawn back as soon as the rooms are finished. There is thus a diagonal line of rooms advancing and another diagonal line, practically at right angles to this, retreating. On the other room entry of the pair, the driving of rooms is commenced at the inby end and proceeds outward. In some instances the entry pillars have been drawn on the retreat as illustrated in Fig. 10, and in others they have been left until all the rooms and room

pillars have been finished. In the latter case it has sometimes been possible to obtain the coal from these entry pillars, but frequently all or part of it has been lost. The method illustrated in Fig. 10 was in use in the Pittsburgh district proper, that is in the high coal along the Monongahela River. Cunningham says that the extraction by this method would average about 80 per cent. Some companies, however, claim an extraction of 90 per cent. Some differences in percentages of extraction may be accounted for by the different methods followed in estimating: the whole bed from the limestone to the top of the seam may be taken into consideration, or the thickness of the slate partings may be subtracted.

One of the principal reasons for taking the rooms turned from one of a pair of butt entries on the advance and those turned from the parallel butt on the retreat was that this procedure made it possible to have the air current always blowing from the room work to the pillar work. This constantly moving current of air prevented gases set free by the pillar work from being carried to men working in advancing places. The miners in the pillar workings used locked safety lamps, while those in the room workings used open lamps.

Until about 1910, mining machines were used in the Pittsburgh district only in room and entry work, while pillar coal was undercut with picks. A method designed to permit the mining of pillars by machines is illustrated by Fig. 11. Cunningham gave the following facts concerning this method:—24-foot rooms are turned on 39-foot centers. After the room is worked out with a machine, a cut about 25 feet wide is made across the end of the pillar; then another cut of the same width is made far enough back on the pillar to leave a stump 5 to 8 feet wide, and the stump is removed by pick work after the machine work is finished. The stumps serve to protect the machine runners and the machine by supporting the top. It was said that at one mine where this system was used 70 per cent of the pillar coal was extracted with machines, and 30 per cent was pick mined. Good falls were obtained, and no ribs were lost. The recovery of timber was not so good as in the Connellsville region or in mines where there is no refuse gobbled along the roadways.*

An old and common method of working is illustrated by Fig.

*For another description of the method in use about 1910 see Schellenberg, F. C., "Systematic Exploitation in the Pittsburgh Coal Seam," *Trans. Amer. Inst. Min. Engrs.*, Vol. 41, p. 225, 1910.

12. Track is laid in the middle of the room, and the room pillars are made as narrow as possible after the room has advanced 100 feet, or to the first cut-through. Commonly no attempt is made to recover the pillar coal beyond this point, though it is often recovered nearer the

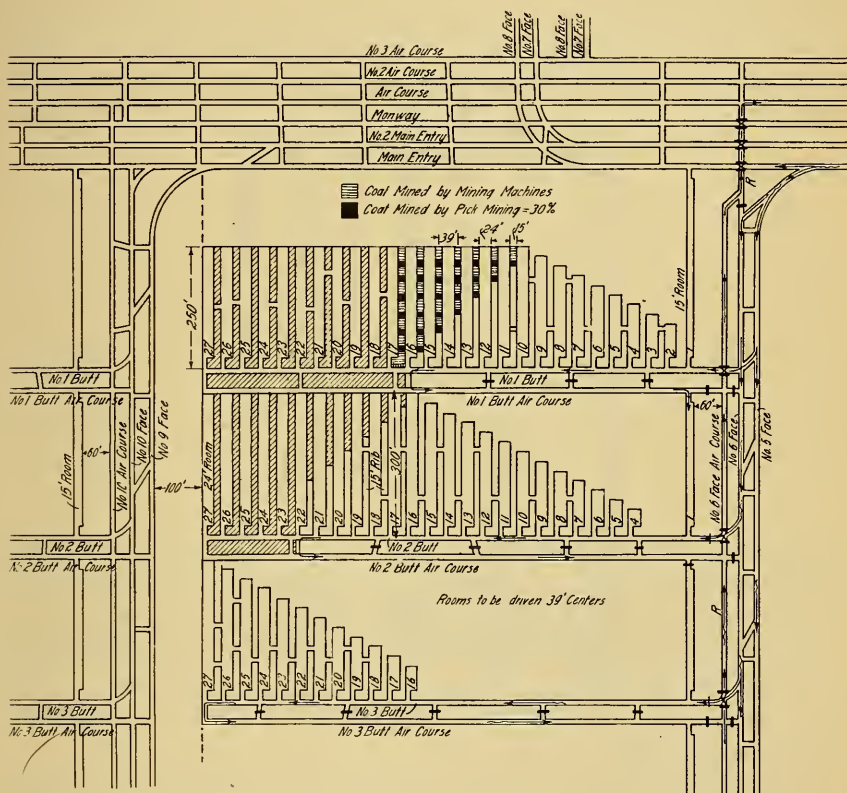


FIG. 11. PILLAR DRAWING WITH MACHINES IN PITTSBURGH, PA., DISTRICT

entries by working the pillar along the side of the fall. One of the chief operators in the Pittsburgh district claimed a recovery of 90 per cent of marketable coal by this method, but it seems that such a recovery could be made only over an area of a few acres and that the recovery over the entire area of the mine would be much lower. Schellenberg expressed the opinion that the recovery would not be more than 60 per cent if an area of 10 acres were considered.

In the discussion of Cunningham's article, G. S. Baton said that entry pillars were not often recovered in the Pittsburgh district except under remarkably favorable conditions. In his opinion not more than 40 per cent of the entry pillars were recovered where there was much

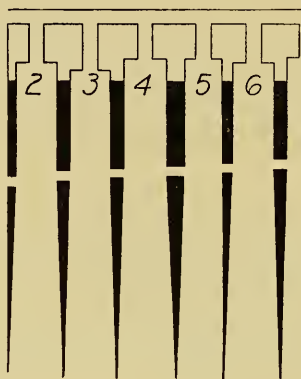


FIG. 12. TAPERED PILLARS

overburden. It is probable that a larger percentage than this is being recovered now in the more carefully operated mines. Another practice in pillar drawing which had been used in the Pittsburgh district and in other districts is illustrated by Fig. 13. A curtain of coal is left to keep out the gob when drawing room pillars, and the loss of coal amounts to about as much as it does where the pillars are narrowed and not drawn.*

It has been seen that in most of the methods employed in the Pittsburgh district, the pillar coal is taken out by pick work, and it has been within only a very recent period that even the room coal has been undercut by machines. Machine mining of pillars is cheaper than pick work and operators have recently introduced this more advanced method wherever it seemed possible. The immediate reason for this action has been the increased cost of production, largely due to high wages and expenses caused by changes in the laws affecting mining, without a corresponding advance in selling price. Since the demand

* Cunningham, F. W., Op. Cit., p. 275, 1910.

is still greater for the lump coal than for the smaller sizes, it has been necessary to increase the size of the pillars to prevent the objectionable crushing of the pillar coal. Another reason for increasing the width of the pillars is to be found in the practical difficulty of using machines on very narrow pillars. The thickness of room pillars varies, but the most common distances between the centers are 33, 36, 39, and 42 feet. With 33-foot room-centers, the room pillars are lost entirely; with 36-foot room-centers, about 55 per cent of the pillars are recovered; and

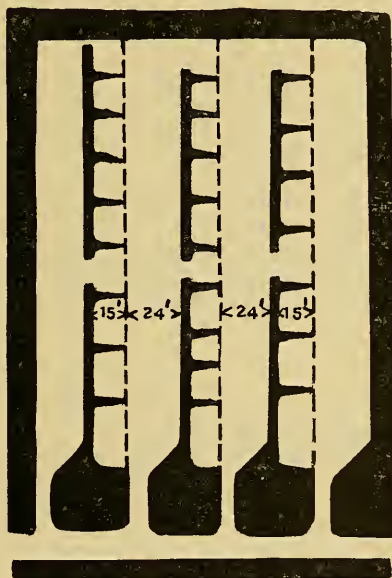


FIG. 13. PILLAR DRAWING, CURTAIN OF COAL

with 39- to 42-foot centers, from 60 to 70 per cent of the pillars are recovered.* Many miners at the present time have not the skill to do the best pillar work, even if the cost were not too high, and for this reason, if a higher percentage of pillar coal is to be won, cutting with machines, which can be operated successfully only on wider pillars, must be employed.

A method adopted for future working at the Marianna and the Hazel mines of the Pittsburgh-Buffalo Company in Pennsylvania and

* Edwards, J. C., and Gibb, H. M., "An Ideal Method of Mining," *Mines and Minerals*, Vol. 33, p. 665; and Edwards, J. C., "Machine Mining in Room Pillars," *Mines and Minerals*, Vol. 34, p. 591.

the rooms are turned on the face of the coal. The butt entries on the side toward which the development of the panel is progressing, that is on the inby side, are termed "advance headings"; the exterior or outby entries are "retreat headings." A chain breast machine is used

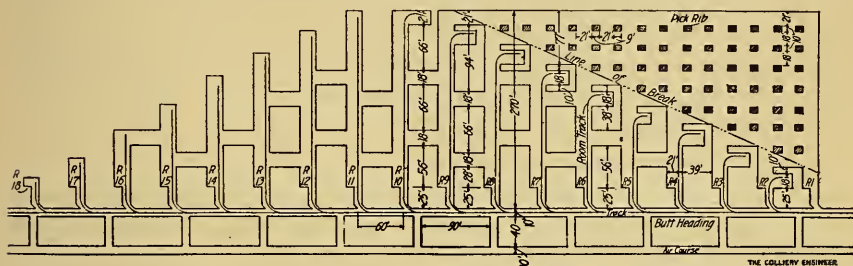


FIG. 15. EXTRACTION OF PILLARS UNDER DRAW SLATE

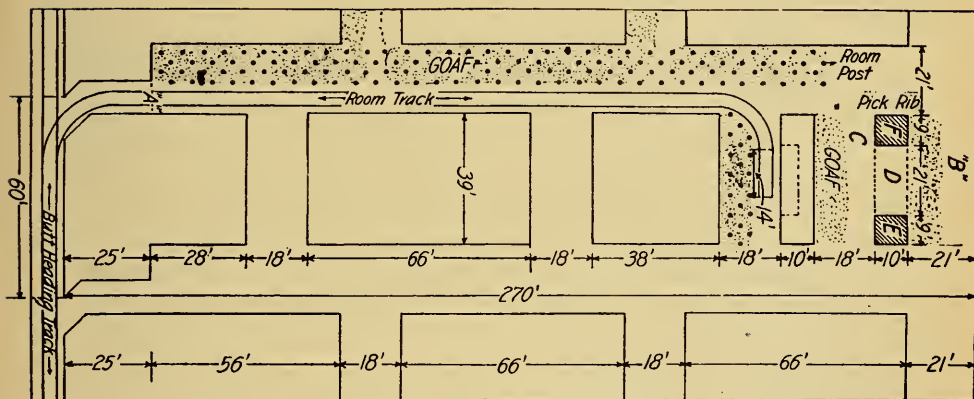


FIG. 16. DETAIL OF PILLAR WORK UNDER DRAW SLATE

in rooms and entries and a short-wall machine on pillars. Both machines continue in use until the last room on the retreat entry is completed; then the short-wall machine is left to finish the pillars, and the breast machine is transferred to another pair of butt entries under development. By the time room 14 is turned, room 2 has been finished, and work can be begun on the pillar between rooms 1 and 2. Rooms on the advance entry are 255 feet long and those on the retreat entry 246 feet long from the entry centers; this difference in length is made, because the chain pillar and entry stumps are brought back with the

room pillars on the retreat entries. The method has been worked out for two general conditions; first, where a draw slate is encountered, and secondly, where there is no draw slate.

The method to be used where draw slate is encountered is illustrated by Fig. 15. In this illustration room 1 is shown as finished. In rooms 2 to 9, inclusive, the pillars are being drawn. Room 10 has reached its limit, and the cross-cut at the face is being driven through the pillar to room 9. Rooms 11 to 18, inclusive, are being driven. Fig. 16 shows in detail the method of recovering the pillars by the short-wall machine where draw slate is encountered. From the point *A*, the track is laid in 14-foot sections, and steel ties are used; consequently, the track is easily assembled or detached. Curved rails are used in the same way so as to give easy access to the cross-cuts. After the cross-cut *B* is finished, the curves and two 14-foot sections of the track are detached, and an 18-foot cut is made in the pillar at *C* by working on the butt of the coal and leaving a stump *D*, 10 by 39 feet. The draw slate from the first cut is gobbed in the room proper. The remainder of the draw slate from this cross-cut is gobbed in the outby part of the cross-cut, and the track is laid in the inby part. A cut is next made through the stump *D*, into the gob above, and small blocks or stumps *E* and *F* are left on each side of the cut to be taken out with the pick. After the block *E* has been removed, all the tracks in the cross-cut, except the two curve rails, are removed; when the block *F* has been removed, the curve rails and the two 14-foot sections of straight track are taken up, and the operation of driving through the pillar is repeated as the illustration shows. In this way the room pillar is extracted back to the point *A*.

The method of operation in the second case, where the draw slate is not encountered, is the same as in the first case, except for the manner of attacking the pillars which is illustrated in detail by Fig. 17. After the room has been completed and the cross-cut *B* driven, the two curve rails and seven sections of track are detached; thus the track is left in position to be assembled quickly for easy access to the cross-cut *H*, which is next driven. The 39- by 94-foot pillar is then split from *H* to *B*, and a 9- by 94-foot pillar is left on each side; then an 18-foot cross-cut *D* is driven through the 9- by 94-foot pillar on the rib nearest the gob, and a stump is left to be removed by the pick. The other 9- by 94-foot block is removed in the same way, and the operation is continued until the whole pillar has been removed. The entry

stumps and chain pillars of the butt headings are won in the same manner as the room pillars.

Both of these methods are in use at several mines and are meeting with success. In both methods 90 per cent of the coal won is cut by machines. This percentage can be increased considerably, since it has been demonstrated that under favorable conditions part of the pick blocks can be recovered by the machine. One-third of the coal is mined from the rooms and two-thirds from room pillars. When draw slate is encountered, the 39- by 10-foot stump pillar will always afford ample

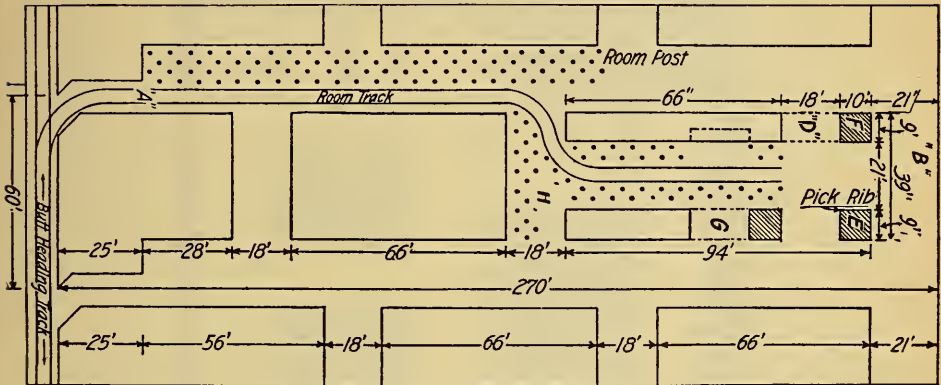


FIG. 17. DETAIL OF PILLAR WORK IN ABSENCE OF DRAW SLATE

protection to the miners; and where no draw slate is found, the two 9- by 94-foot stump pillars together with the timbering will give adequate protection.

Among the plans tried in the Pittsburgh district with the object of reducing the cost of mining by substituting more machine work for pick work is one (Fig. 18) which promised to be successful, but failed, because the miners demanded room-turning prices for the short rooms.* This added cost would have defeated any other advantage of the system. The method, however, seems to be based upon principles which may find application under other circumstances. The method was adopted, because a provision of the mining scale of the Pittsburgh district prohibited the drawing of ribs by machines unless short-wall machines were used. The plan was to continue the employment of the breast machines already in use and thus to increase the percentage

* Affelder, W. L., "Rib Drawing by Machinery," Coal Min. Inst. Amer., p. 232, 1912; and Personal Communication.

of room coal. It seemed profitable to increase the percentage of room coal since the cost for machine-cut run-of-mine coal was 45.28 cents per ton, including a differential of $\frac{2}{3}$ of a cent on account of rolls, while the price for pick mining was 64.64 cents per ton; the average price for cutting, loading, and pick work was 51.92 cents. This average was based on the assumption that the working was regular with rooms 25 feet wide on 40-foot centers. Machine work was done with the common breast machines. The mining system was the ordinary method

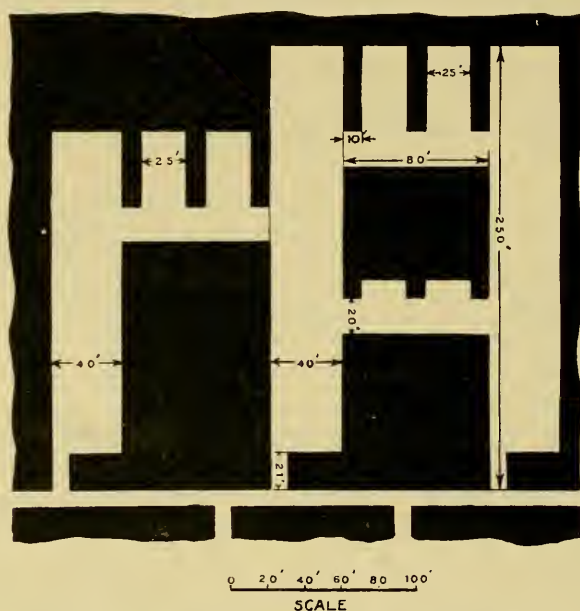


FIG. 18. METHOD OF REDUCING PILLAR WORK IN PITTSBURGH, PA., DISTRICT

of machine mining with 25-foot rooms on 40-foot centers with cross-cuts three runs wide and room necks 21 feet long. Rooms were 250 feet long, but as the neck was 21 feet long, the length of the actual room was 229 feet. All estimations of the percentage of extraction were based on a block 229 feet long and 120 feet wide. In this old system of mining, 65.7 per cent of the coal was produced as machine coal and 34.3 per cent as pillar coal drawn with the pick. The mine was operated on a run-of-mine basis. The new system was started with 36-foot rooms on 117-foot centers; but it was expected that if the system should prove successful, these dimensions would be changed

to 40-foot rooms on 120-foot centers. The illustration shows the form and dimension of rooms. Two 25-foot rooms were driven from the cross-cut with 10-foot pillars on each side. These pillars were somewhat weak, but the rooms were not long, and the pillars were drawn quickly. It was admitted that the system would reduce tonnage for a time if the mines developed were not sufficiently advanced, but it was claimed that when the main rooms had been driven to within 50 feet of their intended length, the production from them would be much greater than from the ribs of three rooms of the older system. As soon as the pillars were drawn, the recovery of the so-called "rooster" coal would rapidly increase the output. In the Pittsburgh bed, the rooster coal lies above the draw slate and a laminated coal 12 inches to 2 feet thick. Although in most mines this coal is not taken out, in the Panhandle district it seems to be better fuel than the bottom coal and is being mined in several places. The cover in the section in which this method was employed was from 75 to 125 feet thick. The small pillars were extracted by pick work, and the rooster coal also was obtained at pick prices.

21. *Connellsville District.*—The methods of production in the Connellsville field have become more intensive than those in the other districts of western Pennsylvania. The excellent quality of the Connellsville coke and the fact that the coal from which it is produced is found in only a limited area have made the coal so valuable that it has been found advisable to pay particular attention to high percentage of extraction. In this district, there has been, moreover, no objection to the crushed coal from pillars as the product goes to the coke ovens where fine coal is desirable. The same fact influenced the relative sizes of rooms and pillars. While in the gas coal district it was thought desirable to take as great a quantity as possible from rooms, in the Connellsville district the practice of getting a large part of the coal from the pillars has developed in order that the percentage of extraction may be as high as possible. It is of interest to trace the more recent developments in mining practice here, because the extraction in the better planned mines of this district is unusually high.

Conditions and methods in this district are discussed by F. C. Keighley* as follows:

* Keighley, F. C., "Mining Coal with Friable Roof and Soft Floor," W. Va. Coal Min. Inst., Dec. 10, 1914; Coal Age, Vol. 7, p. 1008.

“The coking coal known variously as the No. 8, Pittsburgh, or Connellsville seam has in places an extremely bad roof. This difficulty is strongly marked in the Connellsville basin, but can be found in other troughs. . . . The thickness of the coal and its softness might lead one to anticipate that it could be mined cheaply, but the friable roof creates a difficulty which its other qualities cannot overbalance. . . .

. . . “I have projected workings at depths ranging from 200 to 700 feet, using a dozen or more different schemes, and have never found any marked difficulty in protecting the main headings and air-courses of any mine. But I have experienced some trouble occasionally in protecting the branch, or butt, headings, and I have always had more or less trouble with the rib coal in the rooms.

“It is true that in many cases the roof will fall in headings and air-courses in spite of all that can be done, but such falls are gradual and are removed as part of the regular mine operations. On the other hand, when falls occur in the rooms they often come suddenly and cover a large area, and the break extends so far above the coal that they often reduce and may entirely cut off the production from a certain section of the mine, not only for a day, but perhaps for weeks at a time.

“It is clear, then, that any improved method of mining must provide for the protection of the rooms rather than for the care of the headings. . . .

“Panels have been projected 1,000 feet wide and 2,000 to 4,000 feet long. . . . Such a panel is subdivided into a number of smaller panels that are themselves served by two parallel entries driven at right angles or at some other angle to the flat or face heading depending on the pitch of the coal. These are known as butt headings. These sub-panels are generally 1,000 feet long and 300 to 600 feet wide. . . .

“Various widths have been chosen for rooms with 8 to 30 feet as limits, but the general belief is that rooms and headings should be driven 10 feet wide in the Connellsville region. There seems to be no opportunity to improve on this width. The best work has been obtained with large room ribs—50, 70 and 90 feet thick; but the success has not been as great as might be expected, though, with room ribs of the two larger dimensions, and with any ordinary care in mining, a general creep or squeeze cannot occur.

"This is not true when room ribs are made of smaller dimensions, such as 30, 40 or even 50 feet. In the initial stage of rib drawing with such light ribs great success is secured, but when trouble occurs it is usually in the form of a general squeeze or creep that almost paralyzes the output. It has often seemed for a time that the small ribs in the rooms resulted in cheaper mining; but when a squeeze or creep took place the small rib did not permit of the driving of a new road with safety and profit, and consequently the coal remaining in the rib could not be taken out.

"With 70-, 80-, or 90-foot ribs there is always sufficient coal left to permit driving a new road with safety through the pillar no matter

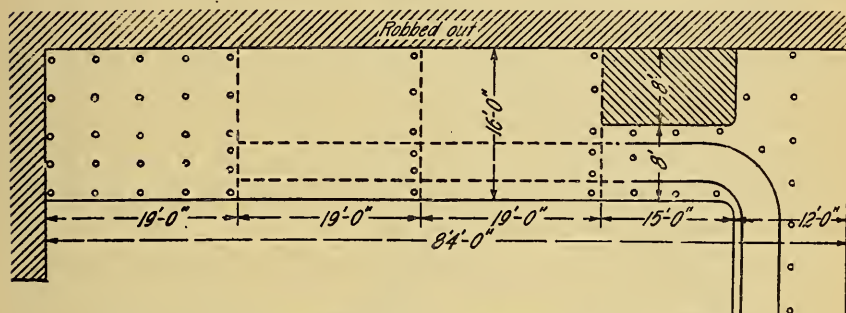


FIG. 19. PILLAR DRAWING IN CONNELLSVILLE, PA., DISTRICT

how badly the roof may have fallen or the coal be shattered on the edges of the pillars.

"Nearly all experienced miners concede that with narrow ribs only 50 per cent of the coal is recovered. The best results claimed is 65 per cent, while 90 and 95 per cent has often been recovered with the larger-rib system. The problem is whether the heavy cost of timber and the still greater cost of labor will counterbalance the loss of from 35 to 50 per cent of coal. I am disposed to believe that the larger rib, making a larger yield possible, will assure a handsome margin."

Fig. 19 illustrates a modern method of pillar drawing.* In this method a cut is made across the pillar, and an 8-foot stump is left; then this stump is taken in a retreating direction in from two to four sections, according to the width of the pillar. In the example given the rooms are 12 feet wide and are on 84-foot centers. After

*Cunningham, F. W., "The Best Method of Removing Coal Pillars," Coal Min. Inst. Amer., p. 275, 1910; p. 35, 1911.

the removal of the coal in each section, the props are drawn and the roof is allowed to fall, the break being controlled by a row of props across the end of the cut. After the last section of the stump has been taken out, the last of the track drawn, and the roof dropped across the room on the line of the end of the remaining pillar, another cut is made across the pillar and the process continues. While the falls represented in the sketches appear to be large, a better break is obtained with these than with short falls. This method has proved to be safer for the miners and to give a greater recovery of posts and coal than the methods which preceded it.

A highly developed and systematized room-and-pillar method is the so-called concentration method used in some of the mines of the H. C. Frick Coke Company,* and developed largely by Patrick Mullen, one of the company's inspectors. This method was designed to satisfy certain requirements, among which were safety of operation, completeness of extraction, reduction of cost through the greater use of machines, and an increase of daily output per man. A patent covering this method has been applied for.

It is well understood that liability to accidents is decreased by close supervision. Under the older system in the Connellsville region, it was possible for the face boss to visit each working place only once in two or three days. In order to increase the amount of supervision without increasing the number of officials, the plan of getting the working faces closer together was tried; this plan, however, necessitated a decrease in the number of working places and in the number of workmen, and it was realized that only by increasing the production of each miner could the output of the mine be kept up.

The only possible way of increasing the output per man was by replacing pick work with machine work. This substitution was made in room work; but it was found that, on account of narrow headings and narrow rooms with large room centers, machines in the narrow work alone would not accomplish the desired results, since the bulk of the coal comes from the pillars. The problem of the use of machines for pillar extraction, which was an entirely new one in the Connellsville district, has been worked out very successfully (see Fig. 20).

* Mullen, Patrick, "New Mining Methods as Practiced by the H. C. Frick Coke Company," Proc. Engrs. Soc. W. Pa., Vol. 32, p. 714, 1916; Coal Age, Vol. 10, p. 700, 1916; Howarth, W. H., "Mining by Concentration Method," Coal Min. Inst. Amer., Dec. 22, 1916; Coal Age, Vol. 9, p. 125, 1916.

The mine is blocked by driving at *A* double butt entries, 10 feet wide on 50-foot centers and 1,200 feet long, across the panel with break-throughs every 100 feet; the pairs of butt entries being driven 350 feet apart, the panel is divided into blocks about 350 by 1,200 feet. These blocks are then subdivided into blocks about 90 by 100 feet, by 12-foot face rooms at *B* 350 feet long on 112-foot centers, driven

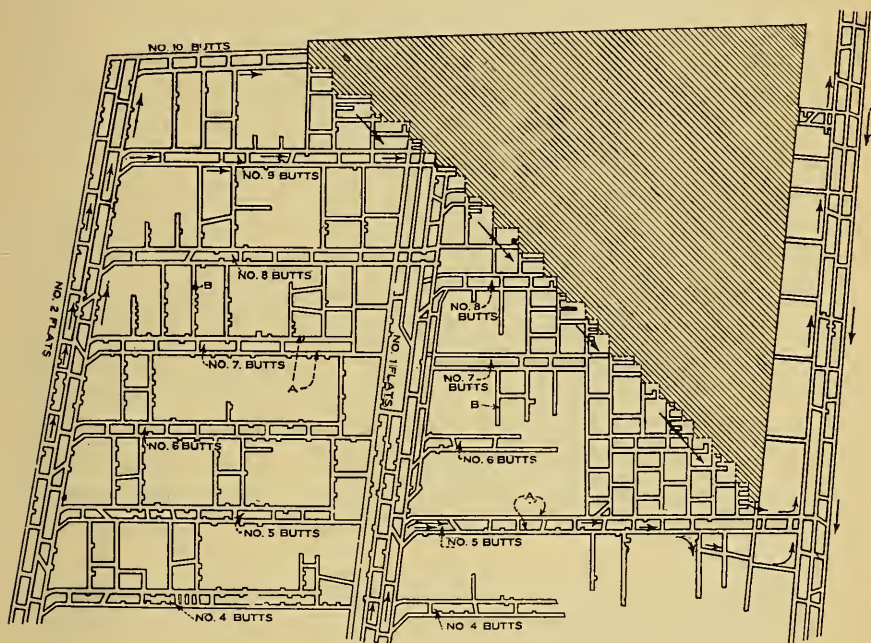


FIG. 20. CONCENTRATION METHOD IN CONNELLSVILLE, PA., DISTRICT

at right angles to the butt entries and connected by 10-foot break-throughs on 100-foot centers. A pillar of this size is considered ample to support any thickness of cover under any conditions of floor or cover to be found in the Connellsville region. In this manner a whole panel can be prepared for the intensive part of the work in which butt rooms are driven from the face rooms 10 feet wide on 25-foot centers.

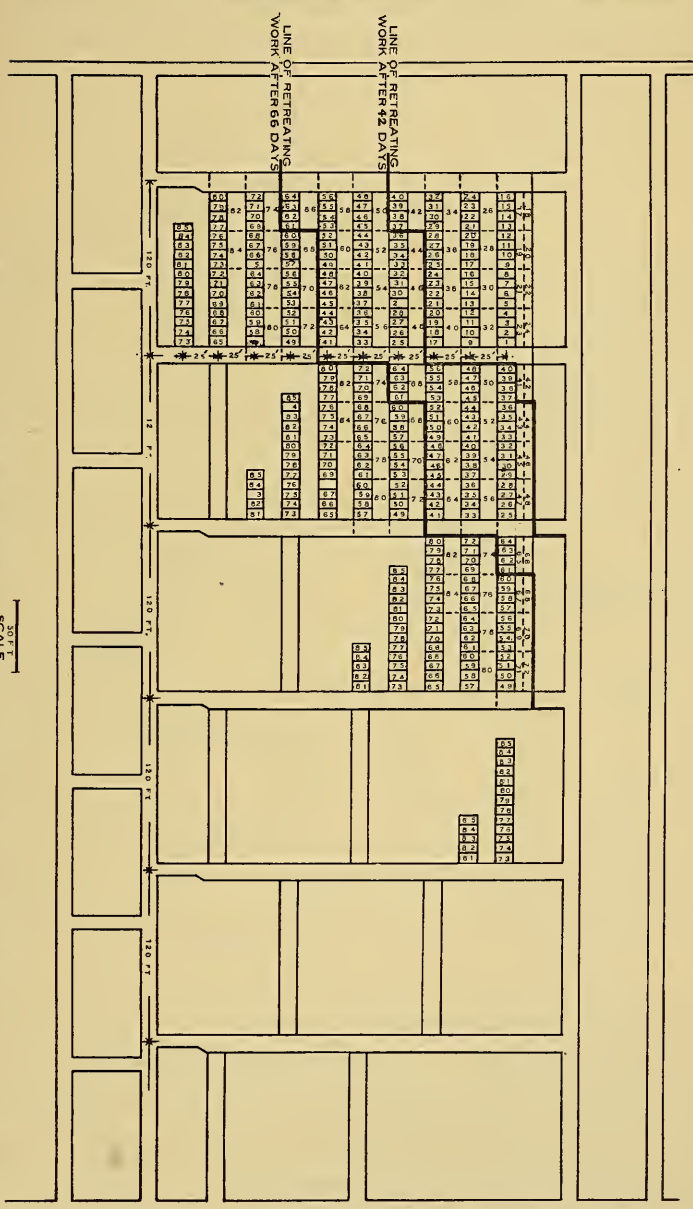
As the main face room advances, the necks of the butt rooms to be driven are excavated to a depth of three machine cuts. After a main face room has been advanced 50 feet, there are available

for the machine to cut two places which allow a production of forty tons; and when the room has advanced to a point where the first cross-cut is turned off, there are three places to cut in each main face room yielding sixty tons. This main room may continue to the end of the section or to the end of the coal field, butts or producing entries being turned off at projected distances. It is necessary that the operation be carefully planned and that the proper order of the work be closely adhered to. Fig. 21 shows the general schedule of operations with the position of the line of roof fracture at different dates. It is claimed that the general plan can be easily modified to suit all conditions such as depth of cover, presence or absence of draw slate, and nature of coal, bottom, and roof.

The projection takes three forms known as (a) maximum, (b) medium, and (c) minimum, according to the rate at which coal is produced (see Fig. 22.) The maximum plan is applicable where the thickness of cover does not exceed 125 feet, where the coal is hard, and where the general physical conditions of roof and bottom are good. The medium plan is applicable where the cover does not exceed 250 feet with the same physical conditions of the coal, bottom, and roof as for the maximum plan. The minimum plan may be applied to coal underlying any thickness of cover; the coal may be hard or soft, and the physical conditions of roof and bottom may be good or bad, provided, of course, that mining machines in any form can be used.

With the minimum plan, the butt rooms are driven in succession so that each room is 50 feet beyond the one succeeding. Two butt rooms advancing furnish 40 tons and one butt rib retreating furnishes 40 tons, or a total of 80 tons on the retreat; the main face room advancing yields 60 tons, or a total of 140 tons from one main face room. These quantities apply, of course, to coal of the thickness of that mined in the Connellsville basin — about 7 feet.

The medium plan will yield the same tonnage from the advancing main rooms, but the retreating work is so arranged that the face of each butt room is 30 feet behind that of the preceding room. This arrangement allows three butt rooms to be advanced at a time with a production of 60 tons, while two butt ribs are being extracted with a production of 80 tons; thus 140 tons are taken from the butt-rooms and ribs and 60 tons from the advancing main rooms, a total of 200 tons for each main room.



In the maximum plan the working of the butt rooms is so timed that the face of one room is 15 feet behind the face of the preceding one; four butt rooms are advanced and four butt ribs are

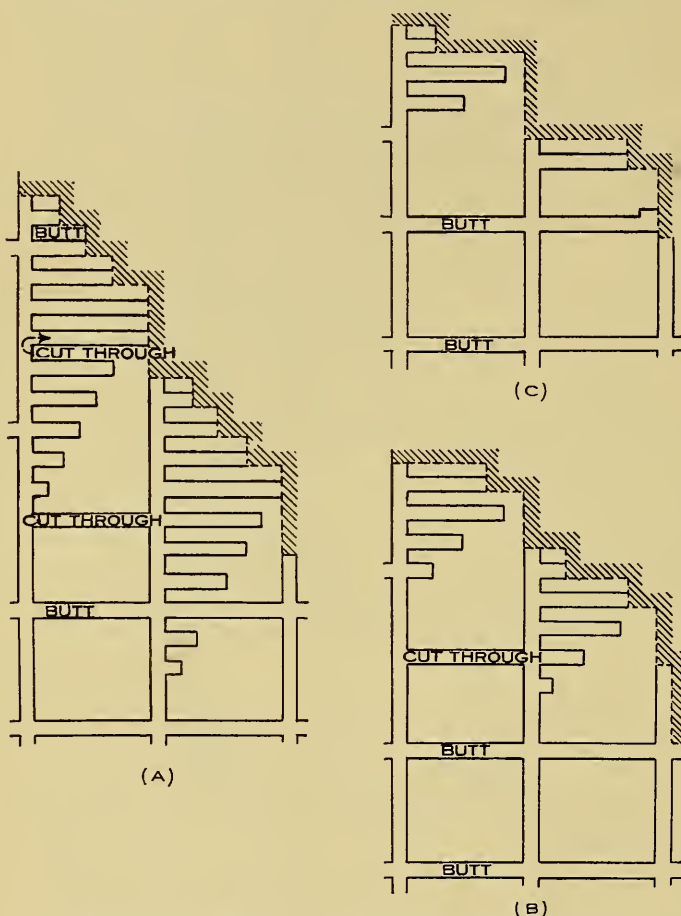


FIG. 22. CONCENTRATION METHOD—MAXIMUM, MEDIUM, AND MINIMUM PLANS

simultaneously withdrawn. The four advancing butt rooms will produce 80 tons and the four retreating butt ribs will produce 160 tons. With the 60 tons produced from the advanced main room, there is thus produced 300 tons for each main room.

The work is thoroughly systematized and proceeds with great regularity. After the miner has cleaned up his place and the day's run

is completed, the machine crew enters and cuts the place to a depth of approximately 7 feet. Following the machine crew, the timber men reset any posts which the machine men have removed, post up any cross bars which have been notched in the coal over the machine cut, and put the place in good condition according to a prescribed system of timbering. The timber men are followed by the driller who bores the holes with an electrically driven drill. The driller is followed by the shot firer who charges and tamps the hole, and after an examination of the conditions, fires the charge. After the coal has been shot down, empty cars are placed by the gathering locomotives so that when the loader arrives at his working place in the morning he finds it in safe condition, the coal ready to load, and the empties in place. Miners loading under these conditions regularly obtain 18 to 20 tons per shift. The average of the loaders for short-wall mining machines in all mines of the company for the month of August, 1916, was approximately nineteen tons per shift. At mines where there is a full equipment of mining machines, the machine coal runs from 80 to 90 per cent of the total output. The recovery under the concentration system is from 90 to 92 per cent, while under the ordinary methods it is 80 to 85 per cent.* In the values given, the top or bottom coal left in place is not considered. The average thickness of top coal left is about 6 inches and the values for extraction, based on the entire thickness of the seam would be somewhat lower than the values given. Coal is left for two reasons. In the entries, from 6 to 8 inches of top coal is left as a protection. In the room work, such top or bottom coal is left in place as is necessary to keep the sulphur content of the coke made from the coal down to the required amount. It is found that the highest sulphur content of the bed occurs at the top or at the bottom and, by frequent analyses, it is determined how much of this top and bottom coal may be left.

22. *Central Pennsylvania.*—A method known locally as the "Big Pillar System" has been developed to meet conditions incident to the soft bottom in the Lower Kittanning, "B," or Miller, bed in the southern and eastern parts of Cambria County, and in the adjoining territory.†

The physical conditions for which this system was developed include a hard roof, very difficult to break, and a soft fire clay bot-

*Dawson, T. W., Personal Communication.

†Silliman, W. A., "Big Pillar System of Mining," Proc. Coal Min. Inst. Amer., p. 76, 1911.

parallel with the entry. There is thus left along the side of the entry a series of blocks 75 by 75 feet or 75 by 100 feet, according to the length of the rooms driven from the entry. As soon as a crooked room has intersected the straight room toward which it is being driven, an intermediate room is turned up the pitch from the crooked room; thus the rooms above the crooked rooms have 50-foot centers. The straight rooms are driven to such distances that the roof will break at the edge of the big pillar or by settling will relieve the strain. Sometimes they are only 250 feet long, and sometimes, under heavy cover, as much as 400 feet.

When the straight rooms are started, they are widened on the outby side so that the cross, or crooked, room can be turned off the straight rib, a matter of importance because of the necessity of storing bottom which is taken up in the roadway. Beyond the crooked rooms, the straight rooms are widened on the inby side; thus the men who drive a room are able to start the drawing of the pillar as soon as the room is finished. The room pillars are drawn back to the crooked rooms, and the irregular little blocks caused by the necks of these rooms are removed as completely as possible. The big block is then left standing to serve as a barrier to protect the entry, and if the space mined out is sufficiently broad, the roof will usually break. Even if the roof does not break, the strain seems to be relieved before reaching the entry. The upper edge of the big pillar may be badly crushed, and the roadway of the room may be heaved almost down to the entry, but the entry itself will be practically unaffected.

When the entry is finished and the stumps are being drawn, the system presents a special advantage in that a better output can be obtained than with the smaller stumps. Where stumps are small, the output is limited to the work of two gangs, but with the big pillars eight or ten places may be worked at all times on the retreat. The big pillars are split by a room driven up from the entry at the same time that a skip is taken along the rib of the old room. These two working places are cut through to the old falls about the same time, and the intervening portions of the pillars are brought back. This method leads to large recovery of coal, although there are no statements available concerning the exact percentage. There is some loss in the extraction of the pillars, and the coal at the edge of the big pillars is badly crushed. The method is not used in other beds in the same district, because the conditions are better.

In the Somerset County district, there has been developed a panel system which permits a high degree of concentration of work and a large percentage of extraction. The coal is low, and the miner is obliged to push his cars to the face of the room and to drop them down to the entry. The seam often dips from two to five per cent, and butt entries are driven off the main haulage slope on a grade of one per cent in favor of the load. From these, entries are driven to the rise at convenient distances, from which rooms are turned on the strike of the bed. Not only does the method result in a high percentage of extraction and facilitate the handling of the cars by hand in the rooms, but it also concentrates the work of mining. Two men working together will produce from ten to twelve tons of pick-mined coal per day, but when the men work singly it has been found that a good miner can load from seven to eight tons per day.* Under this system, the total extraction is reported to be about 93 per cent. About 50 per cent of the coal comes from rooms and entries, and the remainder from pillars.† This method is similar to that illustrated in Fig. 32, page 102, which shows a plan of operation of the Carbon Coal Company in West Virginia.

Because of the recognized objections to the room-and-pillar system, much attention has been given to the possibility of employing the long-wall system in the Pittsburgh bed and in other beds of western Pennsylvania. So far as can be learned, there is only one mine at which a long-wall system is being used in these districts, although there is an approach to it in some so-called "panel long-wall" or "block long-wall" methods. In these methods, dependence, however, is not placed on the weight to break down the coal; in fact, weight at the face is prevented so far as possible by causing the roof to break near the face.

It is worth while to review some of the experiments in the introduction of long-wall methods, because it is only by these methods that complete extraction is attained, although there is a close approach to it in the best applications of some forms of pillar working.

One of the attempts‡ was started about 1899 in the Lower Kittanning, "B," or Miller, bed where the coal was from 3 feet, 6 inches

*Majer, John, "Mining by Concentration Methods," *Coal Age*, Vol. 9, p. 345, 1916.

†Coxe, Edward H., Personal Communication.

‡Claghorn, Clarence R., "A Modified Long-wall System," *Mines and Minerals*, Vol. 22, p. 16; Thomas, J. I., "Mechanical Conveyors as Applied to Long-wall Mining," *Proc. Coal Min. Inst. Amer.*, p. 55, 1907; Delano, Warren, Personal Communication.

to 3 feet, 10 inches in thickness, and had an average dip of eight per cent. The roof was of blue slate and the floor of hard fire clay. Most of the coal lay under a fairly level surface about 174 feet thick, near the top of which was a moderately hard sandstone. The plan adopted involved blocking out the mine with entries and taking out the coal in each entry on the retreat. The faces were 250 to 300 feet

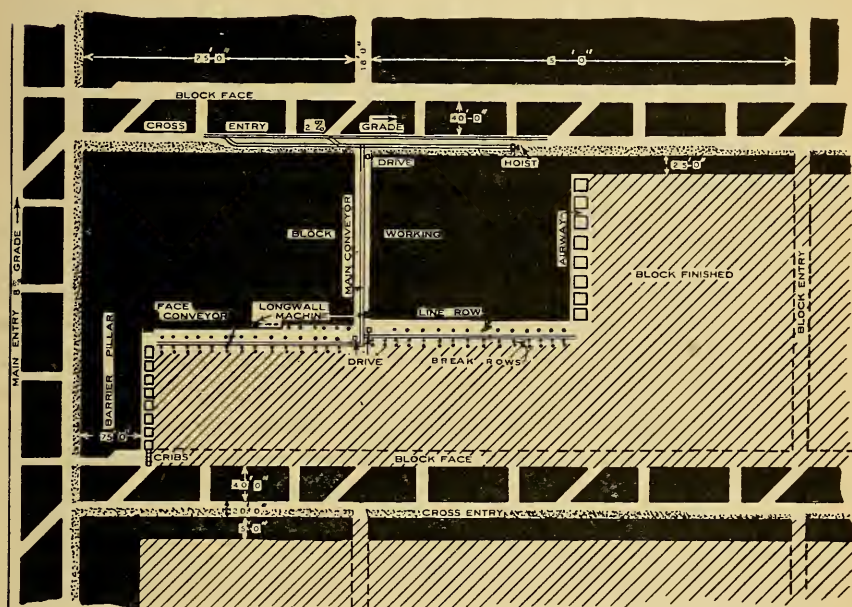


FIG. 24. BLOCK LONG-WALL WITH FACE CONVEYORS

long. One machine-cut produced from 125 to 150 tons of coal. Break-rows were made of stout, round, hardwood posts, 6 to 8 inches in diameter. These were capped with a 2-inch lid of soft wood set on a little slack to facilitate drawing. It was found that the track along the face took up too much space, and a conveyor, which was put in made it possible to set the props closer to the face. At first there were two faces, one slightly in advance of the other, each served by a conveyor which delivered coal to cars let down the block entry. Later, as shown by Fig. 24, a conveyor, by which the coal was lowered to the cars on the level, was installed in the block. Because of the unfavorable trade conditions in 1907, operations could not be carried on

with the regularity essential to the success of the system as then used. The attempt, accordingly, was temporarily abandoned, but a revival of the plan is being seriously considered.

Another attempt at long-wall mining was made with very similar mechanical arrangements in the Cement seam near Johnstown. In this instance two faces were cut, but a shortage of power compelled the abandonment of the experiment before the second face had been completed. Until that time the mining had been economical and the recovery was almost perfect.*

At present the Maryland Coal Company of Pennsylvania is using eight long-wall conveyors at St. Michael. The coal is about forty-two inches thick. No description of the operation is available, but it is evidently considered successful, as the number of conveyors is being increased. Two other companies in Pennsylvania and one in Maryland have recently decided to employ the same method.†

23. *Summary of Facts Relating to the Percentage of Recovery in Pennsylvania.*—The recovery in the Pittsburgh bed, not including the coke district, is estimated to be about 80 per cent. This estimate is made on the following basis: The actual tonnage mined is compared with the computed tonnage of the district worked out; everything between the fire clay and the drawslate is included, and no deduction is made for impurities in the bed or for the average thickness of four inches left on the bottom. This computation is obtained from one of the largest operating companies of the district and is based upon actual measurements. It is the opinion of this company, assuming that this method of calculation is used, that 85 per cent is probably the best possible recovery in this district. Some other companies claim an extraction of 90 to 95 per cent, but this is calculated after deducting the coal left in the bottom, the bearing-in bands, and any other impurities in the bed which are taken out and not weighed. Such high recoveries, of course, imply careful planning for the extraction of pillars and for the execution of this work without delay.‡

Another company the workings of which lie along the Monongahela River south of Pittsburgh, estimates the recovery as 86.7 to 90.6 per cent.

* Moore, M. G., Personal Communication.

† Link-Belt Company, Personal Communication.

‡ Schluederberg, G. W., Personal Communication.

In the Connellsville district the best practice gives from 80 to 85 per cent with the methods ordinarily used there. With the new "concentration" method of the H. C. Frick Coke Company a recovery of from 90 to 92 per cent is obtained.*

In the Johnstown district it seems impossible to obtain estimates of the percentage of extraction, because all the seams in that district vary in thickness within short distances, and are somewhat cut up by rolls. In some seams, for example, the Miller seam in the vicinity of South Fork, the recovery is almost perfect. The conditions are favorable, the roof being well adapted to extraction of pillar stumps in retreating.†

One of the companies operating in Jefferson County claims an extraction of 90 per cent. The operations are in the Lower Freeport bed, and conditions are somewhat peculiar because of bad roof, lack of uniformity of the seam, and faults. Each district requires individual development before an estimate can be made of the proportion of faults to the whole area, and it is impossible to make an accurate estimate of recovery until a district has been completely worked out. The value given represents the proportion of coal extracted from the area mined in which coal existed, and does not apply to the area of faults.‡

One operator in Clearfield County estimates an extraction of 95 per cent, based upon the amount of coal mined up to December, 1916.§

One of the companies operating in Somerset County estimates that, where mines are operated in an area of less than 300 acres and under a cover not exceeding 200 feet, the recovery should be, and in a number of instances is, in excess of 90 per cent. In the case of a property of 1,000 or more acres, where the coal extends underneath a hill giving cover of 300 to 700 feet, the recovery is from 85 to 90 per cent. Low coal, faults, and adverse grades still further reduce this percentage.§

The attainment of the higher percentages in Pennsylvania has been reached only within very recent years, and is not yet by any means universal. There are still in operation a large number of old mines,

* Dawson, T. W., Personal Communication.

† Moore, H. G., Personal Communication.

‡ Van Horn, H. M., Personal Communication.

§ Personal Communication.

§ Delaney, E. A., Personal Communication.

mostly small, in which high percentages of extraction are not obtained. The more recent operations are planned for, and give, probably as high a yield of coal as can be expected from the area worked.

24. *Maryland*.—In the Georges Creek region of Maryland, the Big-vein coal has been mined for about one hundred years, and the methods used there furnish an illustration of progress in coal mining engineering which is especially interesting in view of the increasing attention given to the percentage of recovery.

In a paper on maximum recovery of coal,* H. V. Hesse discussed the wasteful early methods of mining in the Georges Creek region and suggested economic methods, as follows:

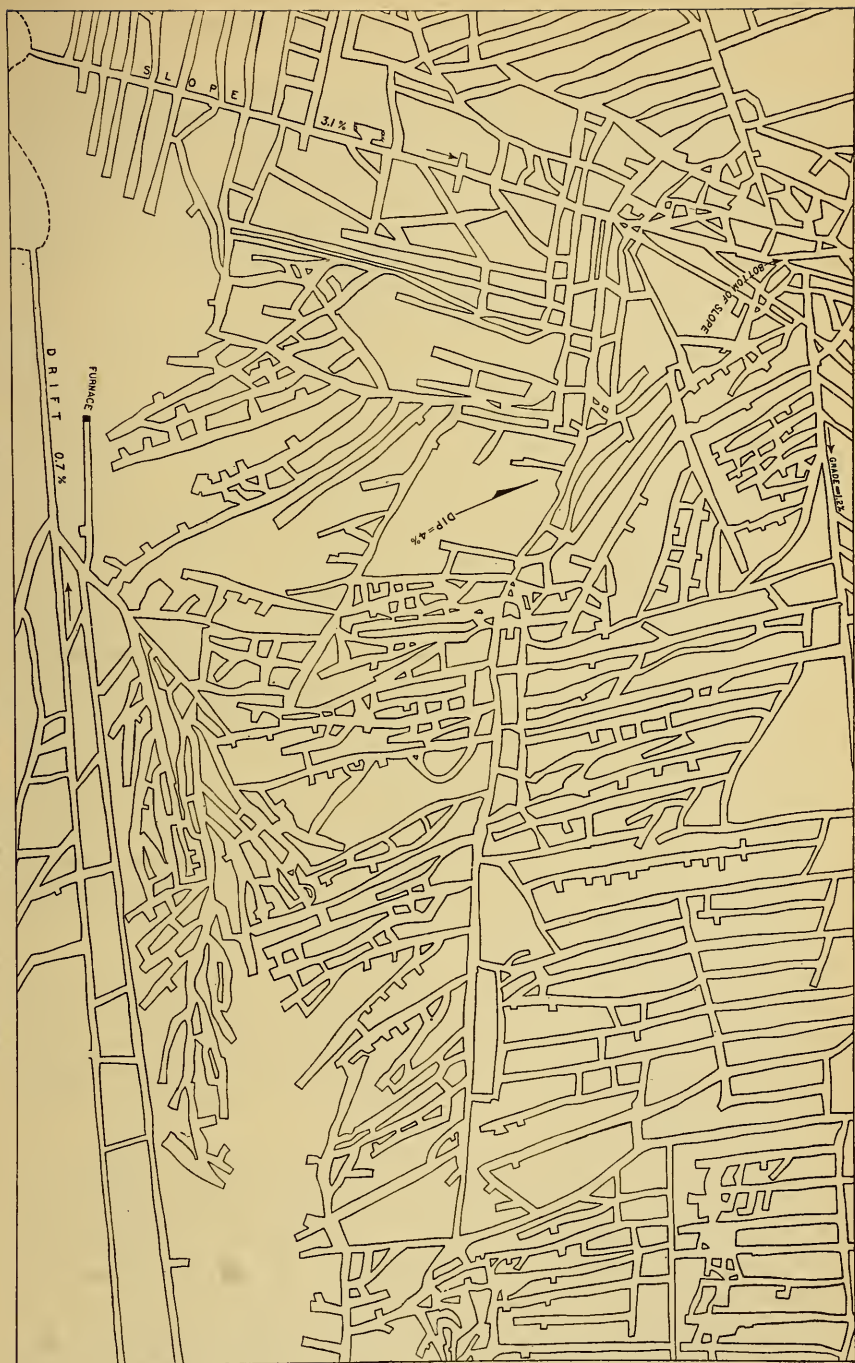
“A region of uniform and unusually severe conditions in the bituminous fields has been selected to illustrate the results obtained over a long period. The Georges Creek region of Maryland, with remarkable deposit of semi-bituminous ‘Big-vein’ coal, has operated in this seam and shipped to the market for nigh unto a hundred years. . . . More than one miner still lives who ‘dug coal’ before the war with the South, and . . . he tells of the detail method of extracting the coal, on account of which thousands of tons lie buried to-day, much beyond recovery. . . .

“The ‘Big-vein’ seam occupies the geologic horizon of the Pittsburgh bed, but differs considerably in structure and quality from the coal of Pittsburgh, Connellsville, and Fairmont. . . . The top coal averaging 2 feet thick is left up for a roof. Where this comes down the strata immediately above promptly follows. Very little of this top coal is therefore recovered. Both roof and breast of the seam contain slips known among the miners as ‘horsebacks,’ which frequently fall out without any warning. The coal is soft and the ‘butts’ and ‘faces’ entirely absent.

“The methods of extraction in vogue at different periods in the history of this field have established the fact that it is impossible to maintain wide working places for any length of time. Headings are driven 8 feet wide and rooms from 12 to 15 feet. In the earlier days there was practically no definite system of extraction, headings and rooms being driven at random and no pillars recovered. Fig. 25 shows such a method in use about 1850. This is reproduced from an actual survey made under the most trying circumstances. . . . It is estimated that fully 55 per cent of the original coal, not counting the top coal, remains and it is expected to recover at least one-half of this, or 27 per cent of the original, by careful operation and the use of about double the amount of timber necessary under a good system of mining. The maximum cover over

* Hesse, H. V., “Maximum Recovery of Coal,” Proc. W. Va. Coal Min. Inst., p. 75, 1908; and Mines and Minerals, Vol. 29, p. 373.

FIG. 25. METHOD OF WORKING THE GEORGES CREEK BIG VEIN, 1850



this district is 300 feet and the few comparatively large pillars, which were inadvertently left standing at irregular intervals, saved the balance from being crushed. In other sections of the same mine where a similar method was followed, but these large pillars not left in, the workings are entirely closed and the remaining pillars, containing over 50 per cent of the original coal, probably lost forever. Fortunately mining operations during this period were not conducted on a large scale and, consequently, the territory thus affected is limited to a very small portion of the company's holdings.

Fig. 26 "illustrates two methods followed during the years between 1870 and 1880. These workings are inaccessible to surveys at the present time owing to the creeps and squeezes induced by the irregular method of robbing the small pillars. . . . In the first method . . . the rooms were 14 feet wide and pillars 26 feet. These pillars were found to be totally inadequate and extracting them impossible. Cross-cutting the pillars at frequent intervals was then attempted after completion of the rooms, but this was generally accompanied by creeps closing a whole district at a time. The maximum height of the superincumbent strata in this territory is 200 feet.

"The second method shown on Fig. 26 was adopted later. . . . By this method headings were driven from the main entry on the rise of the seam at intervals of 1,000 feet to the level above, and two pairs of cross-headings turned to the right. The rooms were driven from these cross-headings at 50-foot intervals and 14 feet wide, leaving a pillar of 36 feet. The length of the rooms varied from 300 feet to 550 feet. These pillars were also of insufficient size, robbing was conducted spasmodically and although more coal was recovered than in the adjoining districts a great deal was lost. In addition to the small pillars, the method of robbing them was calculated to promote squeezes. It appears to have been the method to hold the strata with props until sufficient coal had been removed to enable the weight to break the props. As a general rule, however, before this was attained the weight had induced a creep which is well known to have no limits within a territory of small pillars.

Fig. 27 "represents a method in use in 1890. . . . Rooms were turned as shown from all headings on 100-foot centers and pillars split by half rooms. The length of rooms varied from 300 feet to 600 feet and they were 14 feet wide, leaving pillars $42\frac{1}{2}$ feet wide. These pillars were not strong enough to support the overlying strata of 500 feet and the usual creep resulted when pillar drawing commenced. . . .

Fig. 28 "shows a method adopted in 1900. The maximum dip is 15 per cent and the greatest thickness of superincumbent strata 425 feet. The slope, together with parallel air-course and manway, are sunk on the heaviest dip of the coal and double entries turned off to right and left at intervals of 1,000 feet on grades of $1\frac{1}{4}$ per cent to $2\frac{1}{4}$ per cent in favor of the loads. From these haulways, cross-headings are deflected at intervals of 240 feet at an angle of about 25 degrees and driven on a grade of 4 per cent to 7 per cent. Rooms varying in length from 100 to 800 feet are turned on the rise of the coal from

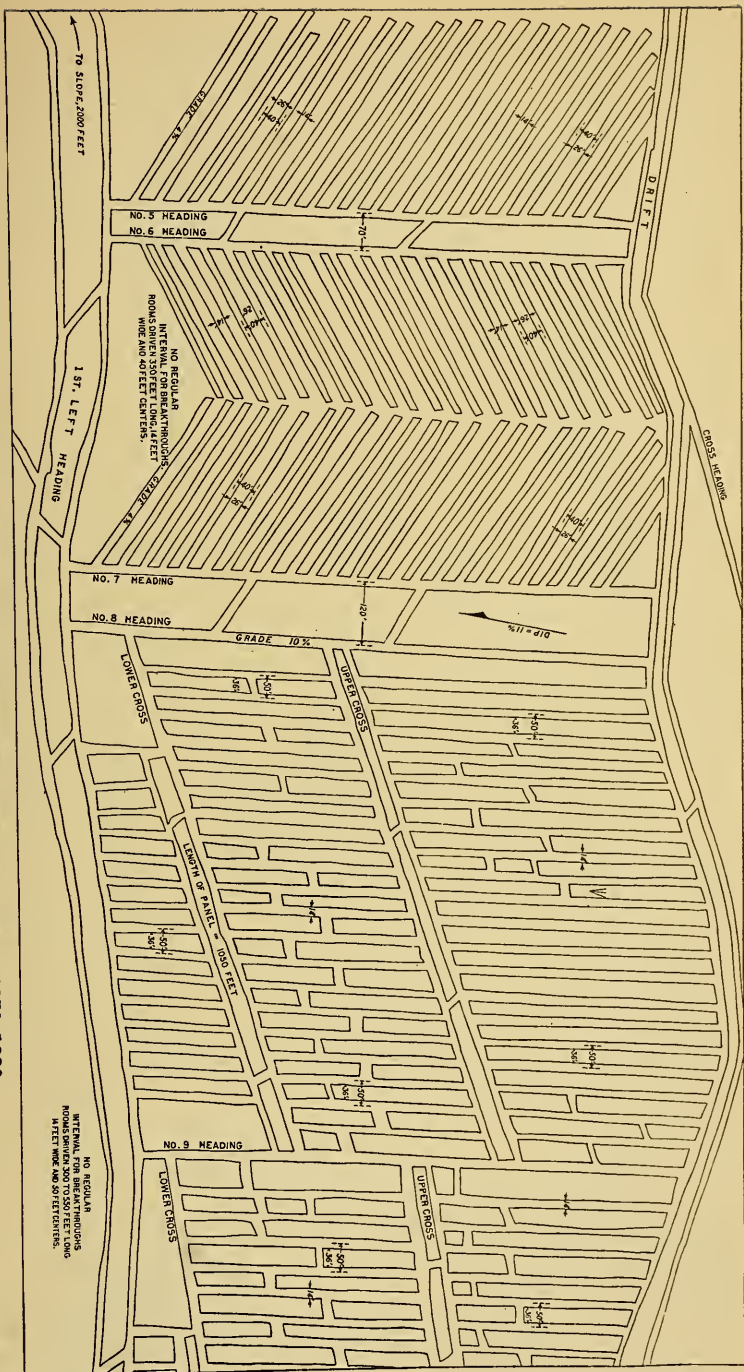


FIG. 26. METHOD OF WORKING THE GEORGES CREEK BIG VEIN, 1870-1880

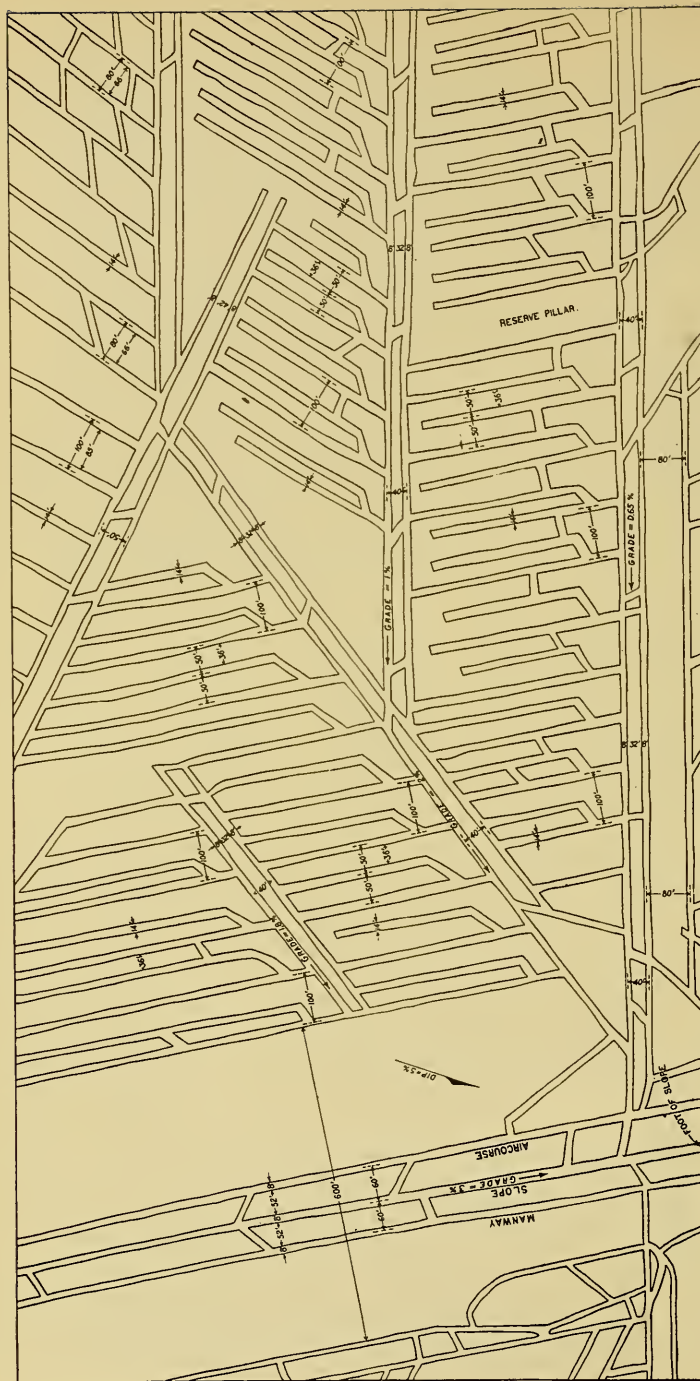


FIG. 27. METHOD OF WORKING THE GEORGES CREEK BIG VEIN, 1890

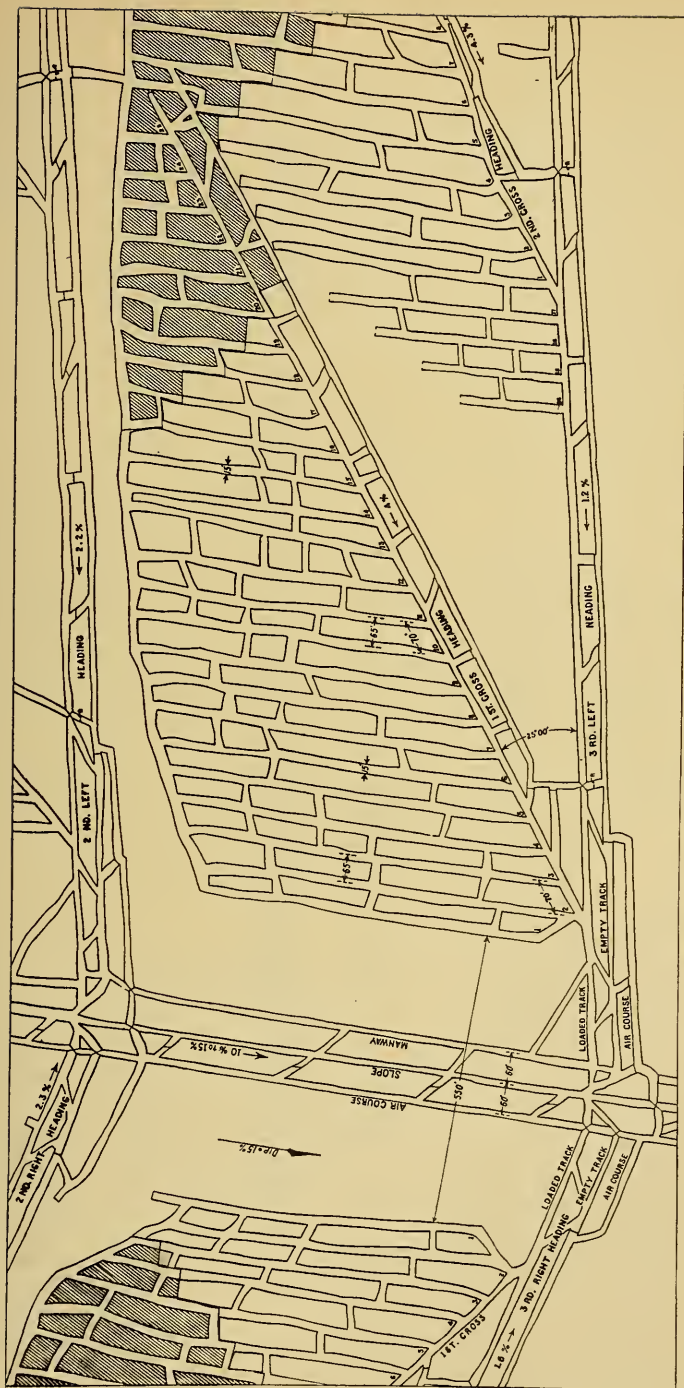


FIG. 28. METHOD OF WORKING THE GEORGES CREEK BIG VEIN, 1900

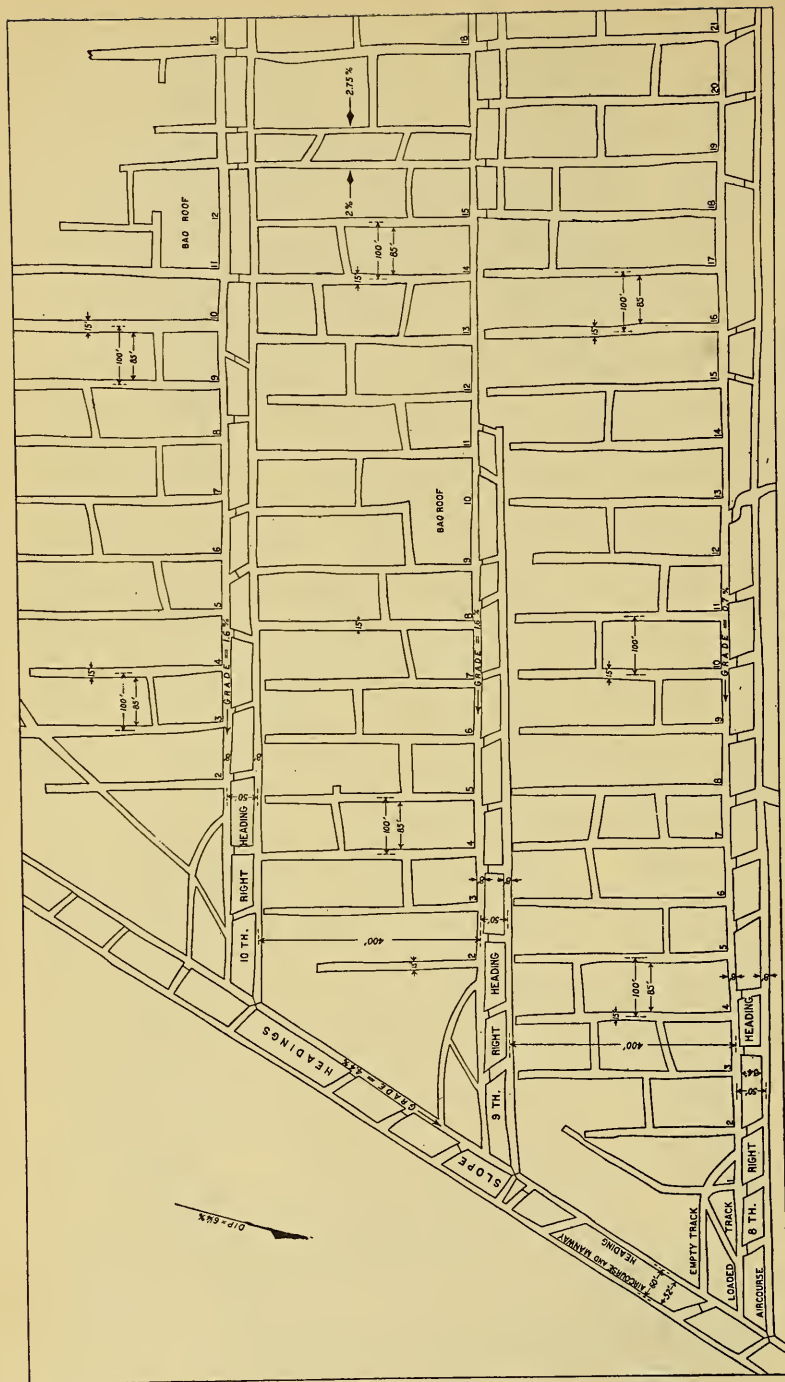


FIG. 29. METHOD OF WORKING THE GEORGES CREEK BIG VEIN, 1904

these cross-headings. The rooms are driven 15 feet wide on 65-foot centers, leaving pillars 50 feet wide. Twenty-five rooms are driven in each of these diagonal panels. Unusually large protecting pillars are left along the main haulage roads. This system has been found to be especially adapted to rapid gathering of cars thus ensuring a large tonnage. It has been found, however, that a very large recovery from the pillars is impossible, owing to the many sharp angles, which, in a thick seam of soft coal, are always difficult and oftentimes impossible to extract. This sharp-angle method was even resorted to formerly in cross-cutting the pillars preparatory to drawing them, but this has been changed to a rectangular method, thereby increasing the actual percentage of pillar coal recovered from 80 per cent to 83 per cent. The distance of rooms apart has also been increased in the last few years to 100-foot centers giving pillars 85 feet thick. It is expected that the extraction of these will show a further increase in the percentage of yield from pillars. The present yield from headings, rooms, and pillars under this system is about 90 per cent, considering the recovery from headings and rooms as 100 per cent.

Fig. 29 "illustrates a method instituted in the latter part of 1904. The main haulway is an extension of the slope from the opposite side of the basin. Double entries are turned off from this entry, on $1\frac{1}{2}$ -per cent grade, 400 feet apart, from which rooms are driven directly on the rise of the coal. Rooms are from 13 feet to 15 feet wide and . . . they are driven at 100-foot intervals, leaving a pillar 85 feet wide. The length of a panel is about 2,500 feet, containing 22 rooms. There are five such panels in this district and when completed it is proposed to draw the pillars in a retreating fashion with the line of pillar work on an angle of 45 degrees across the whole district. A similar method in another district . . . is yielding $88\frac{1}{2}$ per cent from the pillars with a total recovery of 94 per cent from headings, rooms, and pillars . . . the greatest height of the overlying strata is 250 feet."

George S. Brackett states* that in 1898 he made some careful estimates of the percentage of recovery over a period of a year in the Georges Creek region of Maryland. The data for the computations were obtained from two mines which were worked under the following general conditions:

The thickness of coal was 7 feet, 3 inches; the inclination was 5 to 18 degrees; the system of mining was the room-and-pillar retreating method. All the entries were driven to the boundary before any rooms were opened, and a good line was maintained on the drawing of pillars. No. 1 mine had moderate grades, and a better roof than No. 2. No. 2 had grades

* Personal Communication.

as steep as 18 per cent, and the roof was decidedly heavy on pillar workings.

The following results were obtained:

	Total Per Cent of Extraction	Per Cent of Pillars Ob- tained, Including Chain and Barrier
No. 1 Mine	97.6	97.0
No. 2 Mine	82.1	71.3

The average total recovery of the two mines was nearly 90 per cent.

25. *West Virginia*.—The modern methods used in the large mines of West Virginia are among the most advanced found in this country. In many parts of the state, coal mining is carried on by large corporations which are financially able to conduct operations with a view to ultimate economy. In most cases this ability has resulted in the development of methods of operation which lead to very high percentages of extraction.

In the Fairmont district in the northern part of the state, the more recently opened mines are planned for large production and a high percentage of extraction. According to the West Virginia Geological Survey, the Pittsburgh bed, which is the one mined in this district, contains more than 7 feet of clean coal* and the average total thickness of the bed is about 8 feet. The newer mines are projected on the panel system; the last rooms on each room entry are turned first, and the pillars are drawn immediately, a line of break being maintained at an angle of about 45 degrees with the entries. A plan of operation used in this district is illustrated by Fig. 30. The method of attacking pillars is shown in Fig. 31. One of the principal operators in the Fairmont district estimates the recovery, where mines are laid out systematically on the panel system, to be from 85 to 90 per cent of the entire seam.†

A company operating to the south of Fairmont estimates that 85 per cent is a good recovery. In the workings of this company, the rooms represent 27 per cent of the total area; the pillars down to the heading stump, 39 per cent; and the chain and barrier pillars, 34 per cent. The recovery in these three classes of working would be respectively, 100, 90, and 70 per cent. This would give a total yield

* W. Va. Geol. Sur., Vol. II., p. 180, 1903.

† Smyth, J. G., Personal Communication.

of 86 per cent. This company has no very accurate figures on recovery.*

In the mines in the Freeport coal bed in the Piedmont, or Elk Garden, district, there is ordinarily a good shale roof, although there

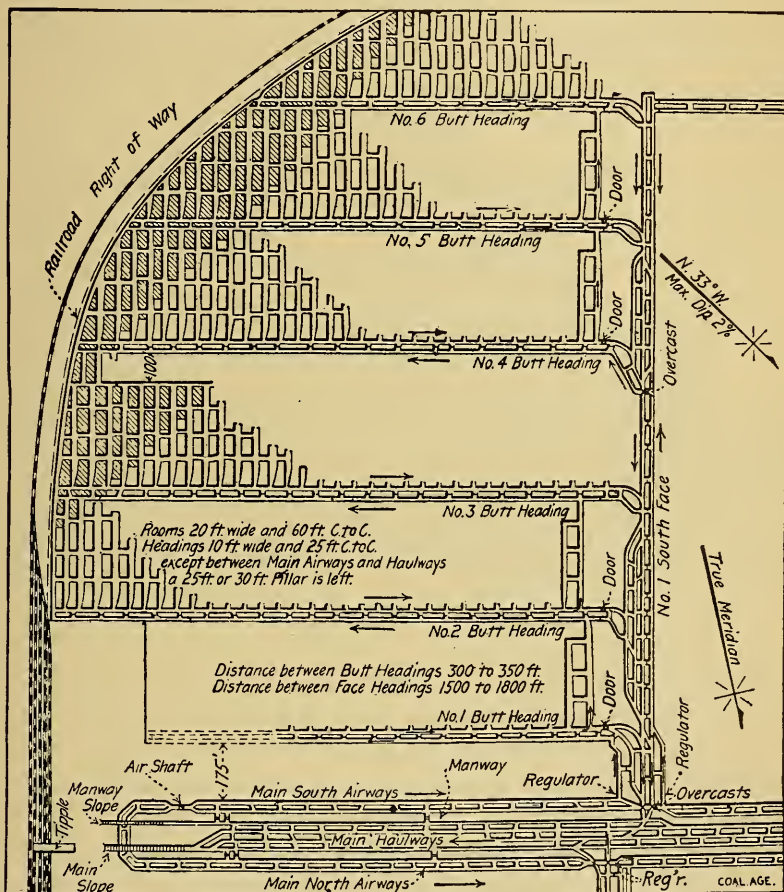


FIG. 30. PLAN OF WORKING—FAIRMONT, WEST VIRGINIA, DISTRICT

are places in which the shale is replaced by sandstone. In the mines in the Kittanning bed there are from 3 to 14 feet of shale above the coal, and above this are 40 feet of sandstone, although there are places where the sandstone forms the roof for short dis-

*Brackett, George S., Personal Communication.

tances. The percentage of extraction in this region is about 90 per cent; in most of the new mines, however, an extraction of 97 or

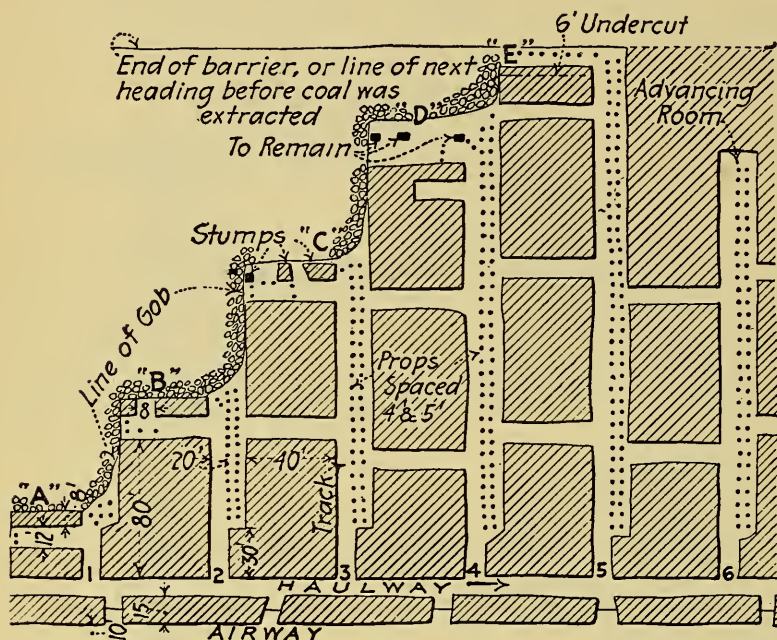


FIG. 31. PILLAR DRAWING IN FAIRMONT, WEST VIRGINIA, DISTRICT

98 per cent is being reached. This rate of recovery is higher than it formerly was, because changes have been made in the order of driving rooms and drawing pillars. The earlier custom was to drive long entries, from which rooms were turned on the advance. The room pillars could not be drawn until the entry was finished because of the danger of squeezes, and as this process sometimes occupied four or five years, falls occurred which made it impossible to recover a high percentage of the pillars. Under the present system, room entries are driven long enough for 20 rooms, and the inside room is turned first. Work on room pillars is commenced as soon as rooms 19 and 20 are finished, and room and entry pillars are taken out rapidly. Nearly all the pillar work is done with picks, and there is little machine work carried on in the district. The pillars are attacked by cross-cuts, and a stump about four feet wide

is left next to the end. This stump is removed as soon as the cut through the pillar is finished.*

In the Central West Virginia district the operations are in the Lower Kittanning bed which averages $6\frac{1}{2}$ feet in thickness.† The conditions are almost ideal for a high rate of recovery. The bottom consists of hard shale. The immediate roof is of bone coal 3 to 10 inches thick, above which are shales of varying hardness, 10 feet to 15 feet thick. Above this layer occurs sandstone of an average thickness of 10 feet, and above this, shale and overlying earth. Nowhere is the overburden greater than 90 feet in thickness. The extremely favorable nature of the roof is shown by the fact that a complete break is easily obtained with as few as 3 or 4 rooms. In more recent workings 20-foot rooms are turned on 50-foot centers. The rooms are driven 300 feet long, and a 50-foot pillar is left between the heads of the rooms and the adjoining air-course. This pillar is never pierced except in case of extreme necessity. The 30-foot room pillars are taken out by driving cross-cuts through them every 30 feet retreating. The entry pillars are taken out with the room pillars. If the entry stumps and the barrier pillars at the ends of the rooms are considered, it is estimated that 75 per cent of the coal obtained is taken out as pillar coal and 25 per cent as room coal. The cost of room and of pillar coal is about the same. Bischoff estimated that the recovery is at least 90 per cent, and possibly 95 per cent, although no accurate records have been kept. In view of the unusually favorable conditions, it seems probable that this estimate is correct as there is no apparent reason for loss, except that represented by the small amount of coal which the loaders fail to shovel up. This statement, of course, refers to only the area of actual mining operations.

The Pittsburgh bed is being worked also in Braxton and Gilmer Counties about 75 miles south of the workings just mentioned. While the same method of working is followed, the physical conditions are different, and the extraction is not more than 75 per cent. The coal is 6 to 8 feet thick. The bottom is of fire clay 8 to 15 feet thick, and the immediate roof is of fire clay 2 to 6 feet thick. Above this occurs a sand rock thicker than that found in the neighborhood of Elkins, with a heavier overburden. In this southern district the

* Personal Communication.

† Bischoff, J. W., Personal Communication.

coast of pillar coal is about three cents greater per ton than the cost of room coal.*

The Kanawha region is unlike the fields farther south in that

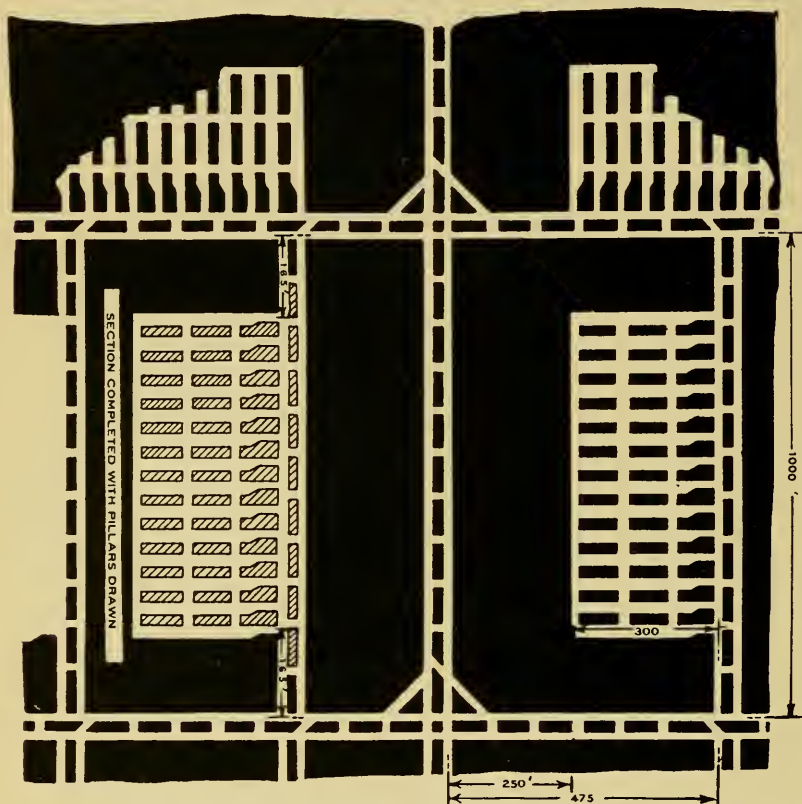


FIG. 32. WIDE BARRIER PILLARS AND ROOM STUMPS, KANAWHA DISTRICT, WEST VIRGINIA

there is a larger number of operating companies with a corresponding lesser concentration of ownership. Because of the number of individual operations it is impossible to give any general or standard method, but the room-and-pillar method is universally used. At least one operator is leaving a large barrier pillar, Fig. 32, and a large room stump for entry protection. The first break-through is driven about 80 feet from the entry, and break-throughs are kept

* Bischoff, J. W., Personal Communication.

perfectly lined up. Rooms are driven in order, and room pillars are drawn back to the first break-through as soon as the adjoining rooms are completed. The recovery under this method is said to be about 90 per cent and where the roof is extremely good as much as 95 to 97 per cent.*

In a paper on the removal of coal from the No. 2 Gas Seam in the Kanawha district, J. J. Marshall reported a very high percentage of recovery and gave the following facts concerning the seam:† The coal bed is made up of two benches separated by a solid parting the thickness of which varies from 10 inches to 40 feet. It has been found that it is not economical to remove this slate when its thickness is more than 24 inches. The aggregate thickness of the two benches averages about 9 feet, the upper bench ranging from 4 feet, 6 inches to 5 feet, 6 inches of clean coal and the lower bench from 3 feet, 6 inches to 4 feet of clean coal. Where it is impossible to mine both benches together, only the upper bench has been taken. The thickness of cover varies from a few feet to 100 feet. After the ordinary method of driving rooms and of drawing pillars on the advance, the mine described has been developed until it is now in position for the butt entries to be worked on the retreating system. On June 30, 1911, the percentage of recovery was said to be as shown by Table 6, nearly all the coal being mined by pick work:

TABLE 6
PERCENTAGE OF EXTRACTION IN KANAWHA DISTRICT

	Total Acres	Percentage of Recovery
High Coal, Both Benches	84.61	91.8
Upper Coal, Upper Bench Only	67.87	98.7
	152.48	94.9

Computations of areas are made from the mine map, and the method of computation does not insure the accuracy of the percentages given. It seems that the values are too high. If there is a loss of 4 feet or more in thickness across the working face, it is recorded, but if the loss is less than this, it is too small to show on the map which is drawn to the scale of 100 feet to the inch, and the recovery

* Cabell, C. A., Personal Communication.

† Marshall, J. J., "The Removal of Coal from the No. 2 Gas Seam in the Kanawha District," Proc. W. Va. Coal Min. Inst., p. 303, 1911.

is regarded as practically complete. It was said to be seldom necessary to record a loss, especially in the upper coal.

In the Cabin Creek portion of the Kanawha district little attention has been paid to the extraction of pillars until recently, and the extraction has amounted to only about 50 per cent. For the past 3 or 4 years, however, the extraction has been about 85 per cent. Since the proper sizes of pillars are now known and the men have a better understanding of pillar work, it is expected that the percentage of extraction will show some further increase.*

Two mines in the New River field, one in the Fire Creek bed and the other in the Sewell bed, have a recovery which is considered practically complete.† At those two particular mines the roof conditions are very favorable; in other sections of the field where they are not so good and where less attention is paid to recovery, it is thought that a fair average extraction is about 90 per cent.‡

The Pocahontas district, in the southern part of West Virginia, is one of the most important coal producing regions in the country, largely because of the high quality of the coal. Pocahontas coal is low in volatile matter and therefore is nearly smokeless; it contains little ash and little sulphur, and it makes an excellent coke. Because of these characteristics there is large demand for it. It is extensively used in coke production, in power plants, in the navy, and in domestic heaters. For coking purposes, however, Pocahontas coal is not used so extensively at present as it was a few years ago. Several beds are being operated, but the principal mines are in the Pocahontas No. 3 bed. The seam varies in thickness from about 4 feet on the west to about 10 feet on the east, but the change is gradual and the thickness is quite uniform within the area of a single mine. This seam has a fire-clay or slate bottom, and a draw-slate roof.§ It always has one streak of bone about 2 inches thick to which the coal adheres on both sides; consequently when a piece of bone is thrown out, about twice as much coal is lost.

The No. 4 bed, which has two streaks of similar bone, is found 75 to 80 feet above the No. 3. Above this bed occurs a seam of interstratified coal and slate locally termed a "black rash." This rash contains on the average about 25 per cent of ash, and it is considered

* Keely, Josiah, Personal Communication.

† Personal Communication.

‡ Cunningham, J. S., Personal Communication.

§ Eavenson, H. N., Personal Communication.

worthless, but sometimes over rather large areas there occurs in it a streak of clean coal from 6 to 8 inches in thickness. Miners are supposed to leave all this rash in their working places, and most of it is left, but sometimes some of this clean coal is loaded out. This fact will explain the higher yield from the No. 4 bed. In other words, the coal actually mined and loaded sometimes has a thickness greater than that considered in calculating the contents of the bed. The fact that a considerable amount of good coal is lost with the bone explains the smaller yield in mines in the No. 3 bed. It is believed that if it were not for this loss the extraction would average about 95 per cent.

As a rule, throughout southern West Virginia, the coal lands are held by land-holding companies, which lease to operating companies. The royalty is generally 10 cents per ton of coal and 15 cents per ton of coke, with a yearly minimum.

The beds are nearly flat and quite regular, of an almost ideal mining height, generally with good roof and bottom, and with little gas and water in the drift mines. It is possible, therefore, to lay out a definite plan of mining in advance, and to follow such a plan more closely than in sections where natural conditions are less favorable. In many cases the landholders specify that the coal shall be mined in accordance with certain plans, and prescribe a minimum of extraction. Certain departures from the standard methods, however, are permitted where it seems advisable.

Fig. 33 illustrates the plan of development formulated by the Pocahontas Coal and Coke Company,* which may be carried out by one of three possible procedures as follows:

Panel No. 1.—Drive rooms on 3rd cross entry as soon as come to, begin robbing as soon as second room is completed and rob advancing on 2nd and 3rd cross entries to within 100 feet of 2nd cross entry, on 1st cross entry drive last room first and rob retreating as shown, taking out the barrier pillar left on 2nd cross entry.

Panel No. 2.—Drive entries to the limit before turning rooms except as shown, turn last room on 3rd cross entry first, begin robbing at inside corner of panel, develop rooms only fast enough to keep in advance of robbing and bring robbing back with uniform breakline until completed to barrier pillars.

* Stook, H. H., "Pocahontas Region Mining Methods," *Mines and Minerals*, Vol. 29, p. 395. Stow, Audley H., "Mining in the Pocahontas Field," *Coal Age*, Vol. 3, p. 594, 1913.

Panel No. 3.—Continuous panel, drive entries to the limit before turning rooms except as shown, turn last room on 1st cross entry first, begin robbing as soon as second room is completed, develop

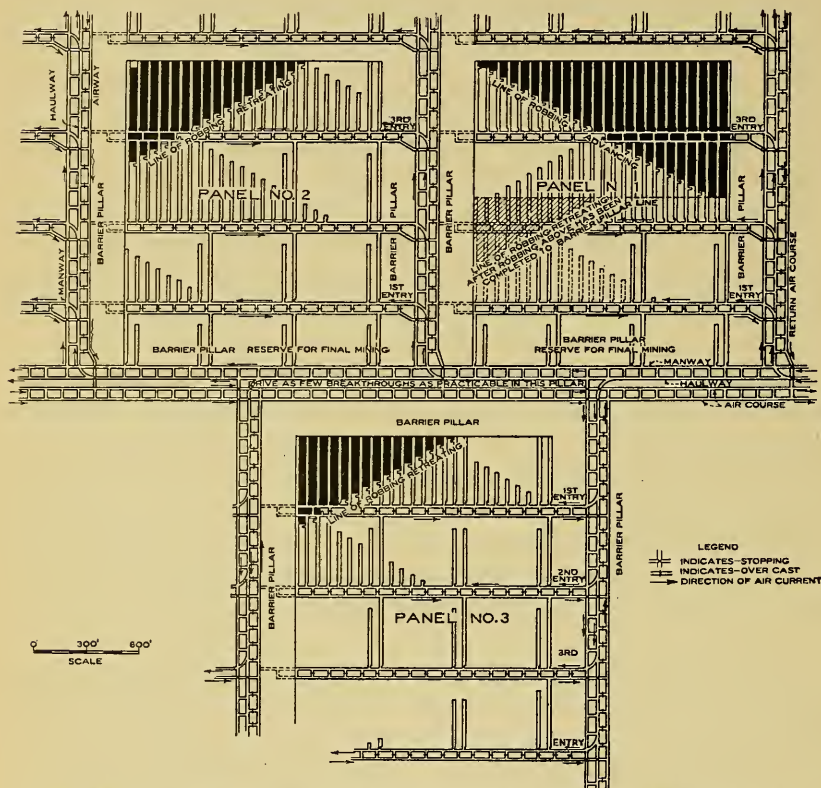


FIG. 33. PLAN OF WORKING OF POCAHONTAS COAL AND COKE COMPANY

rooms only fast enough to keep in advance of robbing, and bring robbing back with uniform breakline until limit of mining is reached.

According to W. H. Grady, chief mine inspector of the Pocahontas Coal and Coke Company,* the essential advantages of this plan of mining include: provision for tonnage during the development period, provision for meeting the market demand, large barrier pillars insuring against squeezes and rendering impossible the de-

* Grady, W. H., "Some Details of Mining Methods with Special Reference to the Maximum of Recovery," W. Va. Coal Min. Inst., Dec. 1913; Coal Age, Vol. 5, p. 156, 1913.

struction of coal over an extended area, 4-entry system for all extensive main entries with two as intakes and two as returns with breakthroughs intervening only at the points where the cross entries turn off, rendering unnecessary the building of expensive masonry brattices every 80 feet, and insuring the maximum quantity of air for ventilation at a minimum cost for brattices and ventilating power, and cross entries with narrow chain pillars which permit the rapid advance of the entry.

The success of this method, with regard to high output, is shown by the values given in Table 7, taken from Grady's article. The percentage of recovery is based on the thickness of the total seam, including the portion rejected. The lower percentages of extraction shown in Table 7 were reached where pillars were being robbed after standing for many years. Operations had not proceeded far enough at the date of the paper quoted to permit a definite statement as to how great the final recovery would be. A later statement* has been received to the effect that the mines of the lessees of the Pocahontas Coal and Coke Company will probably show an average recovery of 90 per cent. Since some of these operations have extended over many years and since they were not so well managed formerly as at present, it is probable that the recovery now is more than 90 per cent.

In the Pocahontas district, much attention has been given to the subject of recovery, and the values which have been supplied by operating companies are as accurate as such values can be made. H.

TABLE 7
RECOVERY OF COAL IN MINES OF POCAHONTAS COAL AND COKE COMPANY

Plant	Thick- ness of Seam in Feet	Acres of Entry Mined	Acres of Rooms Mined	Acres of Pillars Mined	Total Acres Mined	Total Tonnage Mined	Tons Mined per Acre	Theoret- ical Tons per Acre	Per- centage of Re- covery	Pro- portion of Seam Reject- ed
1	6.15	3.06	4.57	11.03	18.66	165,254	8,856	9,922	89.3	0.24
2	5.65	4.40	4.80	14.80	24.00	188,391	8,185	9,115	89.79	0.22
3	5.16	2.68	6.52	15.80	25.00	180,386	7,215	8,325	86.6	0.22
4	4.42	5.88	8.65	13.09	27.62	192,437	6,960	7,131	97.6	0.23
5	5.94	7.00	10.09	19.20	36.29	334,005	9,203	9,582	96.0	0.22
6	4.32	2.11	3.64	9.20	15.04	94,427	6,278	6,969	90.0	0.31
7	5.34	3.31	6.34	0.00	9.65	83,000	8,601	8,614	99.8	0.20
8	5.42	3.72	6.06	9.72	19.50	144,769	8,181	8,777	93.2	0.20
9	4.65	8.10	16.80	2.34	27.24	201,044	7,380	7,534	98.0	0.18
10	8.03	5.20	8.47	10.09	23.76	262,975	11,068	12,923	85.6	0.23

* Eavenson, Howard N., Personal Communication.

N. Eavenson, whose communication has already been referred to with regard to the character of the beds mined, says that the measurements of areas worked out are as close as it is possible to get them on a large scale. The thicknesses given are those of the clean coal, and do not include any bone or black rash. In many instances a record has been kept of coal left in small areas, and the values shown by these tests agree very closely with those given in Table 8. It will be seen from the table that the amount of extraction for mines 9, 10, and 11 is given as more than 100 per cent. This record is explained by the statement previously made concerning the loading out of coal supposed to be left in the mines. At No. 9, the rash is much cleaner than at the other mines of the company, and while the seam is thicker, it carries only a very small amount of dirt; thus a higher percentage of clean coal is given.

TABLE 8

STATEMENT OF THICKNESSES AND RECOVERIES, ALL MINES, UNITED STATES
COAL AND COKE COMPANY 1902 TO 1916, INCLUSIVE

Mine No.	Area Worked Out per Cent		Average Thickness Clean Coal	Net Tons Recovered Per Acre Foot	Percentage of Recovery	No. of Seam Worked	Date of First Shipments
	Rooms and Entries	Pillars					
1	45.5	54.5	5.52	1746	97.0	3 & 4	1903
2	59.7	40.3	5.67	1790	99.4	4	1902
3	61.2	38.8	4.56	1429	79.4	4	1903
4	57.3	42.6	6.19	1581	87.8	3	1904
5	63.5	36.5	6.88	1606	89.2	3	1904
6	63.6	36.4	6.09	1769	98.3	4	1903
7	66.9	33.1	6.27	1770	98.3	4	1905
8	70.8	29.2	5.98	1728	96.0	4	1905
9	74.2	25.8	7.24	1908	106.0	4	1908
10	85.1	14.9	5.31	1806	100.3	3 & 4	1907
11	73.3	26.7	5.26	1807	100.4	3 & 4	1907
12	69.8	30.2	8.22	1622	90.0	3	1908
	65.9	34.1	5.95	1738	96.5		

At the No. 3 mine, which shows the lowest percentage of extraction, the roof is exceedingly bad. Above the coal there occurs a layer of shale and slate, from 5 to 10 feet thick, which it is impossible to support even by close timbering. The mining practice is fully as good at this mine as at the others, but the yield is much less because of the more difficult conditions. The table shows the areas worked out at different mines and the percentages recovered. It is Eavenson's opinion that the average recovery in the larger mines through-

out the Pocahontas field is fully 90 per cent, and that in many mines even a higher figure is reached.

One of the landholding associations of the Pocahontas district furnishes information* concerning some of the operations of its lessees, and submits a table (Table 9) showing the percentage of recovery. It takes account of operations up to January 1, 1917.

TABLE 9
PERCENTAGE OF RECOVERY ON LIVE WORK AND ROBBING
POCAHONTAS DISTRICT
January 1, 1917

AREA MINED			PERCENTAGE OF RECOVERY			
No. of Mine	Live Work, Per cent	Robbing, Per cent	Live Work (Assumed)	Gross Robbing	Net Robbing, Assuming that 37½ per cent of Gross Robbing was Mined as Live Work	Total to Date
1	16.1	83.9	97	84.6	77.2 ¹	86.6
2	.3	99.7	97	73.4	59.2	73.5
3	9.4	90.6	97	81.2	71.7	82.7
4	24.2	75.8	97	80.6	70.7	84.6
5	38.3	61.7	97	76.9	64.8	84.6
6	12.1	87.9	97	87.0	81.0	88.2
7	15.5	84.5	97	77.5	65.8	80.6
8	15.4	84.6	97	79.3	68.7	82.0
9	51.8	48.2	97	83.3	73.9	90.4
10	53.7	46.3	97	83.4	75.2	90.7
11	63.5	36.5	97	86.0	79.4	93.0
12	51.7	48.3	97	81.9	72.8	89.7

¹Values obtained by assuming that 37.5 per cent of the pillar work is done under the same conditions as live work, i. e., with recovery of 97 per cent; thus $0.625X + 0.375 \times 97 = 84.6$, $X = 77.2$.

It will be noted that the extraction at mine No. 2, the first of the leases to exhaust the No. 3 Pocahontas seam, is not more than 73 or 74 per cent. In addition to the losses which will be mentioned, there was a considerable loss here of top coal. A thickness of 18 to 24 inches was left up in the first mining with the expectation that it would be recovered on the retreat, but most of this coal was ultimately lost on account of the bad roof. It is also possible that the loss in the coke yard at this plant, where the maximum number of ovens was run in proportion to the output, amounted to almost double the average tabulated amount.

An inspection of the table shows that, in most cases, the highest percentage of recovery has been reached at those mines where the

* Lincoln, J. J., Personal Communication.

pillar work has been least in proportion to the live work, that is, at those still in the earlier stages of working. Future operations in these mines may be expected to lower these values, but it would seem that 80 per cent would be a very conservative estimate of the average amount of coal that should be won in the various mines of the property up to the exhaustion of all properties under lease.

J. J. Lincoln has discussed losses in the Pocahontas field, and the facts brought out are of general interest in connection with the subject of coal recovery. It is said that losses may be considered under three headings; (a) mixing of coal with refuse, (b) loss in drawing pillars, and (c) loss in coke making.

There is loss in removing the bone and pyrite from the coal, as some coal adheres to the refuse. Under the present mining methods, this loss is from 2 to 4 per cent, and occurs in both new work and robbing.

The following losses are to be expected in pillar drawing in addition to loss of coal attached to refuse: (1) In drawing each stumps are occasionally crushed by the pressure of the top before broken rock from the adjacent gob from covering the coal. This loss will run from 3 to 10 per cent, according to conditions. (2) When the pillars are drawn by splitting, a similar loss, frequently greater, occurs. (3) As the drawing progresses small sections of stumps are occasionally crushed by the pressure of the top before they can be removed. (4) Stumps, sections of pillars, or entire pillars may be crushed by the weight before they can be removed, or may be surrounded and cut off by the broken top. In the mines where actual losses from this source are closely recorded they do not reach 1 per cent, and there is no mine in which they will reach 2 per cent. The third loss is not directly chargeable to mining, but occurs in the making of coke. Where the tonnage of coke produced is used as a measure of the amount of coal taken out, this loss becomes significant in calculating the percentage of coal won. The ratio used by the company in all calculations of tonnage has always been 1.6 tons of coal to one ton of coke. This ratio assumes an actual average yield of $62\frac{1}{2}$ per cent of coke, but in practice this yield is not obtained, the average yield under existing conditions being nearer 55 per cent. This loss has always been charged, with the other losses, directly against the mining. This ratio cannot be used directly in determining the actual amount of coal mined, because only a part

of the coal is coked. The loss varies in amount from 1 to 5 per cent, according to the conditions in the different operations, and the actual percentage of extraction is slightly higher than that given in the table because of this error.

Losses in mining and coke making may then be tabulated as follows:

“(a) Removing Refuse—Coal thrown into gob with bone and sulphur bands . . . 2 per cent to 4 per cent	
(b) Robbing:—	
1. Stumping, or	
Splitting pillars . . . 2 per cent to 10 per cent	
2. Sections or Stumps . . . 0 per cent to 2 per cent	
3. Pillars lost 0 per cent to 2 per cent	
(c) Additional coal consumed in coke making over and above the amount covered by the constant of calculation,	
1.6 tons equals 1 ton coke	1 per cent to 5 per cent
Total . . . 5 per cent to 23 per cent”	

A peculiar system followed by the Gay Coal and Coke Company of Logan, West Virginia, and called a single-room system* has resulted from an attempt to apply the long-wall method to a seam the average thickness of which is 5 feet, 7 inches. This seam dips to the southwest about $1\frac{1}{2}$ per cent, is practically free from partings, and is of the nature of splint coal, the bottom bench being rather strong, and the top bench somewhat friable. The average thickness of cover does not exceed 500 feet while the maximum is less than 1,000.

A block of coal was cut by two entries 600 feet long, Nos. 4 and 5 (Fig. 34). These were connected at their extremities, which were 300 feet apart. The purpose was to take out the block of coal thus formed in a retreating direction by commencing at the inner end and by working outward with a face about 300 feet long. One hundred beech or hickory posts were used to support the roof near the face. The top of each was covered with 1-inch poplar, and the

* Gay, H. S., “A Single-room System,” Proc. Coal Min. Inst. Amer., p. 157, 1906; Mines and Minerals, Vol. 27, p. 325, 1906; Personal Communication.

bottom with 1/16-inch sheet steel. Each post was mounted on a hydraulic head weighing about 700 pounds and tested to a pressure of 3,000 pounds per square inch. These were set along the face 6 feet apart in parallel rows 3 feet apart. The cost of the equipment was approximately \$5,000.

When the walls became more than 30 feet apart, rows of props on 15-foot centers were set 8 to 10 feet apart. When the distance between the walls had reached 60 feet, the portable posts were put into use, a row of 50 posts on 6-foot centers being set 10 feet from

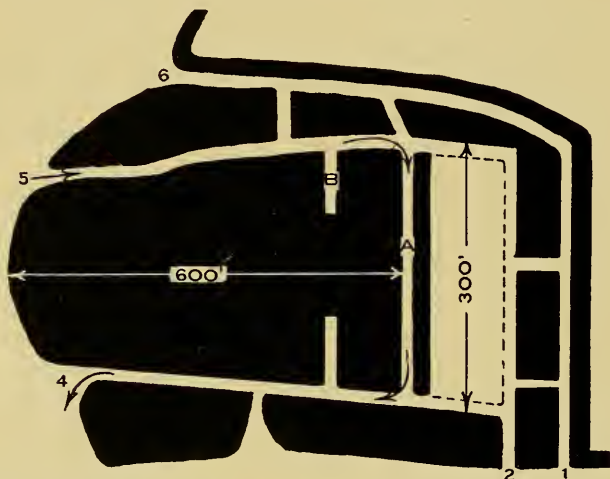


FIG. 34. SINGLE ROOM METHOD, LOGAN COUNTY, WEST VIRGINIA

the face. The heads were covered with wooden cap pieces, and the plungers were raised by a pressure of 50 pounds per square inch. When the face had advanced 6 feet farther, the other 50 posts were put in; as the work progressed, the first row was moved 6 feet ahead of the second, the posts being moved one at a time. An occasional row of posts similar to the first was also set as a precautionary measure.

When the walls were 100 feet apart, it was thought advisable to blast down the roof. The portable posts were set in a single row 6 feet from the face. Examination after the rock had fallen showed that the immediate roof consisted of a seam of strong sand slate at least 30 feet thick without any sign of a parting and that difficulty

would be encountered in attempting to apply the long-wall method in this mine.

The advantages to be derived by working on a long face were so great that the company devised another plan which involved working out the remainder of the block by a system of rooms 80 feet wide, parallel with the long-wall face and separated by 30-foot pillars. Each room was to be opened by driving a sub-entry across from entry No. 4 to entry No. 5 (Fig. 34), and thereafter the manner of working was to be identical in every respect with that of the long-wall system.

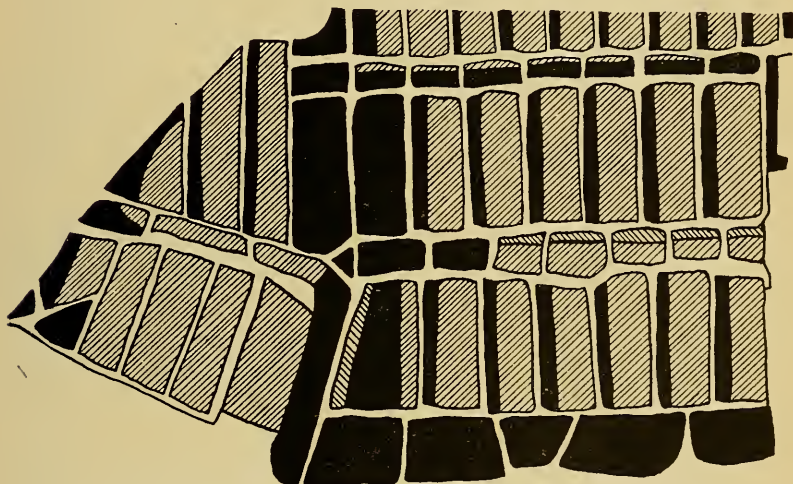


FIG. 35. BIG ROOM METHOD, LOGAN COUNTY, WEST VIRGINIA

From the experience gained it was thought that, with the roof in normal condition, rooms could be worked 90 feet wide with 30-foot pillars. Entries Nos. 4 and 5 were therefore continued eastward, and sub-entries were spaced for the rooms as shown on the map (Fig. 35). The last of the sub-entries was widened to 40 feet, and a single row of ordinary props, 8 to 10 inches in diameter, was set close to the face on 15-foot centers. When the face had moved 10 feet farther, a second row was set. When the room had reached a width of sixty feet, a row of portable posts was set on 10-foot centers, and at 70 feet another row was set. The ordinary props were used for detecting the action of the roof.

One of the advantages of this system of mining is that a large amount of coal per employee may be obtained. In fact the production per man is considerably greater than with the room-and-pillar method, and greater even than would be possible with the long-wall method. The highest rating in this seam for car distribution, exclusive of this mine, is 13 tons per loader; this mine is rated at 20 tons per loader. It is the opinion of Mr. Gay that it would be possible to produce about 11 tons per inside employee per 9-hour day for five days a week. The method also results in the recovery of a very high percentage of coal. A calculation based upon the number of tons shipped and the area excavated, according to planimeter measurement, indicates that the extraction was 85.9 per cent.

Since this description was published, the system has been modified to reduce the narrow work, but the general plan has been followed. Instead of driving a single room to form the working face, parallel rooms separated by an 18-foot pillar are driven; thus the use of brattices is unnecessary, and ventilation is improved. The hydraulic posts were soon abandoned, as posts without the hydraulic heads are cheaper, and they are easily recovered. One of the most important facts concerning the operation is that there has not been a single fatal accident in the mine since work was begun.

Another system in which an effort was made to obtain the advantages of long-wall working was tried a few years ago in West Virginia.* In developing this system (Fig. 36) triple entries are driven from the outcrop, near which double entries are turned off at right angles. From these, entries are driven parallel with the main entry; thus blocks of coal about 900 feet wide are cut off. Block entries are turned from the main entry and from these side entries, parallel with the cross entry, spaced about 500 feet apart, and driven for about 800 feet; thus the coal is blocked into areas approximately 500 by 800 feet. In working these blocks, a room is turned first at the end of the block entries to form a working face for the long-wall machine. The blocks are then worked back toward the main entry for 500 feet; thus a barrier of 300 feet protects each main entry. Track is laid along the face as near the coal as possible, and is moved as the face progresses. The roof is allowed to fall, but the line of break is kept at the correct distance from the face by three or more rows

*James, W. E., "Block System of Retreating Long-wall," Proc. W. Va. Coal Min. Inst., p. 137, 1911; Cabell, C. A., Personal Communication, and Patent Specifications.

of props, the last row being moved forward after a cut is made. The trolley wires are hung on hangers as usual, but to keep them tight they are carried on portable drums. Current for the motor and machines along the face is taken from the trolley wire on the entry

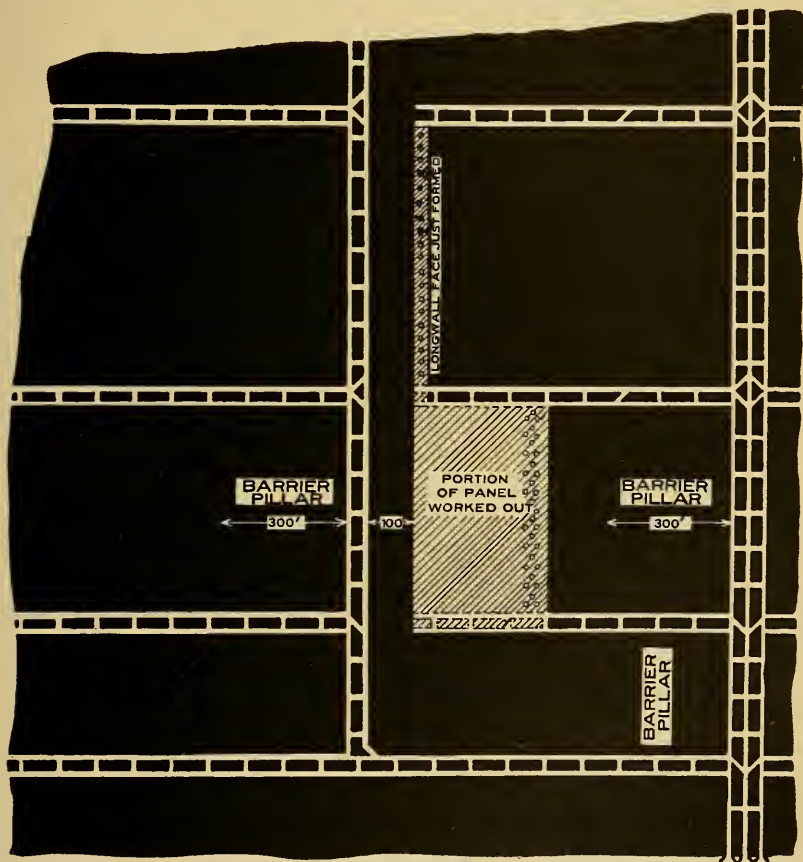


FIG. 36. BLOCK SYSTEM OF RETREATING LONG-WALL, WEST VIRGINIA

by means of a cable which is also coiled on a portable drum. Ventilation is controlled by placing a regulator in the return air course of each block. Each block or face is provided with a separate supply of fresh air by having overcasts placed at the air courses to admit the return air into the main air course entry. No doors are required at any points in the mine.

It was claimed for this method that a block will produce 440 tons per day with the use of only 500 feet of track in addition to that on the entries, that practically all the coal is obtained, and that the work of the rolling stock and cutting machines is concentrated. The trial of this method was temporarily abandoned because of an inadequate car supply, but the work was considered successful. It was not carried far enough to provide reliable data for an estimation of total extraction.

26. *Ohio*.—The literature of coal mining contains little information regarding conditions in Ohio. Some of the operations in the Hocking Valley district have been described in articles which state that large quantities of coal are left in the roof because of the poor quality of the product. An article on Hisylvania Mine No. 23 states* that the bed mined in the Hocking Valley district is the Middle Kittanning, Hocking Valley, or No. 6. The bottom consists of a few inches of fire clay overlying hard rock. The roof is of shale, 6 to 8 feet thick. The coal bed consists of three benches. The thickness of good coal is about 6 feet, and above this is about $5\frac{1}{2}$ feet of a poorer coal separated from the lower portion of the bed by a distinct parting. This upper bed, with the upper bench of the lower bed, is known as top coal. In this district all coal in excess of 6 feet, and in many places in excess of $4\frac{1}{2}$ feet, is to be credited to this upper bench which has a maximum thickness of 10 feet.

James Pritchard† estimates the percentages of extraction in the districts as follows:

In the Pittsburgh vein district, the Cambridge field, and the Hocking field, districts which produce approximately three-fourths of the coal of the state, the rate of extraction will range from 60 to 70 per cent. In the Massillon and Jackson fields, the rate of extraction may reach 85 per cent. In the Deerfield and Mahoning districts, the rate of extraction may reach 85 per cent. Throughout the remainder of the state, the maximum percentage of extraction will run from 60 to 70 per cent. The average rate of recovery is approximately 60 per cent, with a minimum of 55 per cent and a maximum of 75 per cent.

* Burroughs, W. G., "Hisylvania Mine No. 23," Coll. Eng., Vol. 34, p. 421.

† Pritchard, James, Chief Deputy and Safety Commissioner of Mines, Personal Communication.

The average value represents the most common percentage of extraction.

The conditions in the Pittsburgh vein district of eastern Ohio are described by Roby* as follows:

The extraction is limited by various physical and commercial conditions. The roof consists of a bed of weak coal of poor quality, variable in thickness, and separated from the main bed by a layer of drawslate which disintegrates on exposure. Overlying the roof coal is an unstratified "soapstone" 4 to 8 feet thick, and above this is a thick layer of hard limestone. The country is hilly, and the total cover varies from 30 to 600 feet in thickness. The character of the roof makes it necessary to leave larger pillars than would be required under a good roof. The roof coal is left up, and as it is poor in quality and not marketable, it is not considered in making estimates of extraction. The room-and-pillar method is used. There has been much discussion concerning the possibility of applying the long-wall method, but the fragility of the roof and the tendency of all rock below the limestone to shear off at the solid face have seemed to make the method impracticable.

It is desirable that rooms be worked out as quickly as possible because of the tendency of the roof and pillars to fail. Because of these conditions, the numerous interruptions which have been caused by strikes and business depressions have tended to make the rate of recovery lower than would have been the case with uninterrupted operation.†

J. C. Haring states that the recovery in the Massillon district has not exceeded 75 per cent. The highest recovery in the district is probably at the Pocock No. 4 mine where it is estimated that fully 90 per cent of the coal has been obtained.

At Steubenville, the mine of the LaBelle Iron Works has been operated on the long-wall system since 1913; prior to that time the room-and-pillar method was employed.‡ The bed is the Lower Freeport which is a little over 3 feet in thickness and has a good shale roof.

At present a considerable amount of stripping is being done in the No. 8 coal in the vicinity of Steubenville. The coal outcrops on

* Roby, J. J., Personal Communication.

† Haring, J. C., Personal Communication.

‡ Burroughs, W. G., "Long-wall Mining at Steubenville, Ohio," *Coal Age*, Vol. 11, p. 697.

the slopes of hills, and the rapid increase in the thickness of the overburden restricts the operations to a narrow belt, although they may extend for a considerable distance along the outcrop.

27. *Kentucky*.—The development of coal mining in Kentucky has been comparatively recent, and some of the Kentucky fields are still so young that it is impossible to obtain estimates of recovery. The state may be divided broadly into two districts, the western district being closely allied with the fields of Indiana and Illinois, and the eastern district with those of West Virginia and Virginia.

The statements presented in the following paragraphs concerning the western district are made on the authority of N. G. Alford,* who says that the fields contain about 38.3 per cent of the coal-bearing areas of this state and that in 1912 47.7 per cent of the total production came from this district. The smallness of many operations is shown by the statement that 21 per cent of the mines produced less than 10,000 tons per annum each; 51 per cent produced less than 60,000 tons each; 23 per cent produced more than 100,000 tons each; and two companies, operating 18 mines, produced 2,750,000 tons each.

Generally, the rate of recovery in the mines of western Kentucky is about 66 $\frac{2}{3}$ per cent, although in some instances it is as low as 44 per cent. Without an exception the mines of this district are developed on the room-and-pillar system with double or triple entries. With the exception of two or three isolated operations, all the coal is produced from three seams.

Most of the coal comes from the No. 9 and No. 11 beds, the former producing about three-fourths of the total output of the field. This bed is present in eight counties and approaches 5 feet in thickness. In most places it is reached by shafts of 300 feet or less in depth, although there are some local surface depressions which permit access by slopes or drifts. It has a black shale roof and a soft fire-clay bottom.

The No. 11 seam lies from 40 to 100 feet above the No. 9, and follows the latter in commercial importance. Its average thickness is 6 feet. Above the coal is a stratum of limestone of thickness varying from a few inches to 40 feet. This limestone is usually separated

* Alford, Newell G., "Problems Encountered in Kentucky Coal Mining," Ky. Min. Inst., 1913; Coal Age, Vol. 5, p. 674, and Personal Communication.

from the coal by a thin stratum of heavy laminated clay 6 to 24 inches thick. This top adheres uncertainly to the limestone above it, and presents a constant danger. Near the outcrop the top becomes very treacherous. The bottom of this seam consists of soft fire clay which frequently heaves in haulage entries that have been opened for some time. About half the mines in this seam are shaft mines, and the remainder are drift mines.

The third seam in commercial importance is No. 12, which is found best developed in Clay and Webster Counties. Its approximate depth below the surface is 225 feet. Its average thickness is 7 feet. The bottom is of fire clay which is high in calcium and which disintegrates rapidly when drainage water is directed through it in ditches. The roof consists of light gray disintegrated shale 10 to 15 feet thick. If all the coal is removed, this top will fall to a height of 6 or 8 feet, and heavy timber sets, thoroughly and solidly lagged, are required to support it. Because of this condition it has been found necessary to leave 16 inches of top coal as a roof. Sixty per cent of this top coal is recovered from rooms, but no attempt is made to recover it from entries. When the development of the No. 12 seam was begun, rooms were driven 21 feet wide on 33-foot centers; but this width of pillar was found to be too narrow, and it has been increased to 20 feet, with rooms 21 feet wide. Under these conditions a recovery of 44 per cent is the best which has been reached up to this time. This low percentage of recovery is in part due to physical conditions, and in part to over-development and keen competition.

Several factors contribute to limiting the recovery in the No. 9 and No. 11 beds. The most important of these is probably inadequate planning of future workings. Frequently pillars are left too small; consequently the bottom heaves, and the pillars are crushed. Partial recovery of pillar coal by taking slabs off the ribs is not general, and the total recovery of pillars has not been attempted. The operators in this district hold the opinion that pillar robbing in the No. 11 seam is particularly hazardous and impractical, because heavy limestone overlies the seam; in the Connellsville district of Pennsylvania, however, pillars are successfully drawn under a heavy limestone. Alford expresses the opinion that if the workings in the No. 11 seam were properly laid out and started, little difficulty would be found in increasing the percentage of recovery.

In this district there is an over-production with a limited market; consequently competition is keen. When business conditions are normal, the margin of profit is so small as to preclude any costly improvements, and little money has been expended in experiments. Another source of loss lies in the waste at the tipples in the summer, because the demands of consumers are more exacting in times of dull markets. Good coal attached to lumps of pyrite is often discarded in large quantities, and so far this waste has been accepted as unavoidable. This district furnishes one of the best illustrations of the effects of over-development and lack of harmony of interests. There are, however, some operations which are carried out on a considerable scale and with careful attention to the proper planning of the work.

The estimates covering production mentioned previously are confirmed by a personal communication from another operator, S. S. Lanier, who has been a close observer in the district for thirty years and who estimates that the extraction is about 65 per cent.

The eastern Kentucky district is of such recent development that estimates of production are not very reliable. H. D. Easton, operating in the southeastern part of the state, thinks it safe to say that a recovery of 90 per cent is being reached in the Straight Creek seam in Bell County, but much trouble has been experienced from squeezes due to lack of systematic working.

Mines are operated on the room-and-pillar system with rooms turned from both sides of the room entries. Rooms are generally 40 feet wide with 20-foot pillars. Cross entries are driven about 1,200 feet apart, and these usually extend to the property line or to the outcrop. It has been the practice to extract the pillar coal on the retreat, and so far as possible to keep the face lined up over a sufficient distance to get a fall of roof. Room tracks are swung across the face of the pillar and are moved as the pillar is drawn back. If the pillars are narrow, the room tracks are not moved even though the coal has to be shoveled 15 or 20 feet.

There has been no very systematic work in pillar recovery in the southeastern part of the state, and pillars or stumps have been left scattered promiscuously, with the result that many costly squeezes have occurred. One company has lost an entire mine as a result of this practice. It has been the general opinion that it would be impossible to get a clean break in the overlying strata because

of the solid sandstone above the coal, but such breaks have been obtained very successfully, even in rather limited areas.

The Harland district, the Hazard district, and the Elkhorn district are all too newly developed to provide a reliable basis for estimating recovery, but it is possible that the percentage of recovery will be very high.

In eastern Kentucky the surface is of no great value, and it is almost invariably owned by the coal companies so that the necessity for sustaining it is not a factor affecting the percentage of extraction.

There are some operations in the central southern part of the state, but the mines are not yet sufficiently developed to yield adequate data for reliable estimates of extraction.* The mines are opened by drifts, and as the coal is irregular, the hills have been entered at many points. A heavy sandstone occurs between the two seams worked, and pillars have not been drawn in the lower seam, because it has been feared that the upper seam would be damaged. In the upper seam, pillar drawing has not been practiced to any great extent, because when tried, it has resulted in breaks extending to the surface through which considerable water has entered. It is the intention of the operators to extract the pillars when the mines have been worked out, and the final percentage of extraction will probably be high.

28. *Tennessee*.—Little information is available on the percentage of recovery in Tennessee, and the statements obtained are not altogether in agreement. One operator,† formerly connected with the industry in Tennessee, states that a few years ago the mining practices were not good. On the first mining, about 50 per cent of the coal was taken, and the ultimate recovery was probably about 80 per cent. Because of the low value of coal lands, less effort is made to get a maximum recovery than in some other districts where coal lands are more valuable.

R. A. Shiflett,‡ Chief Mine Inspector, says that it would be difficult to give any general percentage for extraction since the coal measures vary in dip from horizontal to 40 degrees, and in some

* Butler, J. E., Personal Communication.

† Coxe, E. H., Personal Communication.

‡ Personal Communication.

cases from horizontal to vertical. In a large number of mines it is impossible to map out any definite method of mining, and conditions have to be met as they are encountered.

Nearly all the drift mines are developed on the double-entry room-and-pillar system. Rooms are driven from 250 to 300 feet, and room pillars are drawn as soon as the rooms are finished. If conditions are favorable, the entry pillars and room stumps are recovered on the retreat; and where the coal is practically level, 40 to 50 inches thick and with good roof and bottom, about 90 per cent is extracted under careful management. This percentage is not reached in many mines because of lack of attention to high extraction. It is thought that the extraction in general does not exceed 65 per cent, but that this percentage could be greatly increased by proper methods and careful management.

One company,* whose method was to turn rooms on the advance and immediately to draw room pillars back to within 65 or 70 feet of the entry on completion of the room, obtained a recovery of nearly 90 per cent up to about the beginning of 1916. At that time four cross entries were lost from heaving of the soft bottom. The cover is 500 to 800 feet in thickness, the coal is 56 inches in thickness, and the bottom is of soft fire clay from 4 to 7 feet in thickness. This company is planning the introduction of the long-wall method. A face of about 300 feet will be formed by connecting the ends of two entries. It is thought that the single stick timbering, with perhaps an occasional crib, will be sufficient. It is expected that the bottom will heave and reach the roof as the latter bends down. The scarcity of labor and the irregularity of the car supply make the success of long-wall operations somewhat doubtful; and if it is necessary temporarily to abandon this method, another which is illustrated in Fig. 37 will be adopted. In this method apparently a little more than 50 per cent of the coal would be taken out from rooms, and the ultimate percentage of extraction should be almost complete. The method will permit concentrated working, and much of the trouble due to the conditions of the floor and roof will probably be avoided.

29. *Alabama*.—Although Alabama† is an important producer

* Hutcheson, W. C., Personal Communication.

† Strong, J. E., "Alabama Mining Methods," *Mines and Minerals*, Vol. 21, p. 195, and Personal Communication.

of coal, the working conditions are not so good as those of Pennsylvania, Kentucky, and some other states. One of the distinctive features of the Alabama coal field is that although there are five seams of coal, rarely more than one of them is workable at any one place; in one portion of Jefferson County, for instance, the Pratt seam, which is considered the topmost workable seam of the Alabama coal measures, may have a working thickness of 4 feet, while at a distance of two or three miles the same seam may not be more than

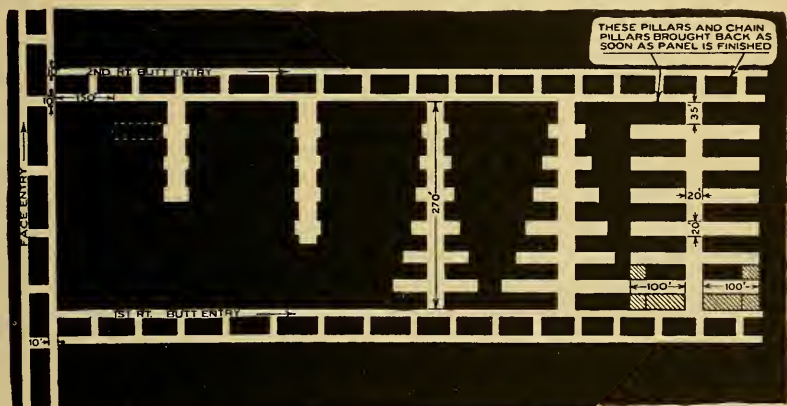


FIG. 37. PROPOSED PLAN OF WIND ROCK COAL COMPANY, TENNESSEE

2 feet thick. The coal beds, including the Pratt bed, vary rather abruptly within a few miles with regard to thickness, impurities, and character of roof.

The Pratt seam has been worked longer and more extensively than any other seam in this district. It is probable that the recovery from this seam, under the best methods of working with the room-and-pillar system, is about 87 per cent. This percentage applies only where the thickness is three feet or more; coal thinner than this cannot profitably be removed.

The Mary Lee seam, which is supposed to contain the thickest workable coal, lies about 300 feet below the Pratt seam, and is the one that the operators of the Birmingham district expect to work during the next twenty years. So far as it is known, the thickness of the bed ranges from 6 to 10 feet. It is difficult to get reliable estimates of the recovery of coal in this seam, but one operator reports,

on the basis of an experience of twenty years, that the recovery has been about 90 per cent.

There are mines on lower seams in other localities, but little attention has been paid to the extraction of a high percentage of the coal. The operation of these mines depends largely upon market conditions, and they are probably not operated, on the average, more than half the time. Under these conditions the loss of pillar coal caused by falls of roof is necessarily large.

The Alabama mines, with the single exception of the Montavallo mine, where a change is being made to the long-wall system, are worked on the room-and-pillar system. The larger operations are in the neighborhood of Birmingham, and in this district the carboniferous measures are tilted and broken to a great extent. This condition affects the roof of the coal under cover for a considerable distance. Both top and bottom are of variable character.

In the larger operations at least, the triple entry system is used. Commencing at a distance of 800 feet from the surface, cross entries are usually driven about 350 feet apart. Until the entries have been driven a few hundred feet, it is not possible to determine whether they should be narrow or wide enough to provide storage for the impurities of the bed and the brushing of the roof. Rooms are generally opened narrow also, (30 feet wide) with 25-foot room pillars, until it is determined whether the character of the overlying strata and of the floor will permit the working of wider rooms.

Probably 75 per cent of the large operators in the district have adopted the plan of immediate pillar drawing in preference to that of driving the narrow work to the limit and pulling the pillars upon the retreat. In a number of instances, rooms are driven 40 feet wide with 30-foot pillars and are worked for a distance of 300 feet, or to the entry above; then a cut is taken across the end of the pillar, and the pillar is drawn back to the entry stump. When the room pillars are drawn on the advance, there is no difficulty in getting room stumps and air-course pillars after the entry work is complete.

Strong estimates the recovery in mines operated by the larger corporations to be from 87 to 90 per cent. Priestly Toulmin, another operator,* confirms these values by stating that the average extraction in Alabama is not less than 75 per cent and not more than 80 per cent, so that possibly 77½ per cent would be a fair value. In

* Personal Communication.

some mines the extraction is less than 60 per cent, and in others it is more than 95 per cent.

C. F. DeBardeleben* furnishes the following estimates:

At one operation where 768.6 acres had been worked over, assuming the average thickness of coal to be 4 feet and that 25 cubic feet of coal in place make a ton, the coal available was 5,356,834 tons

Tons mined 3,028,960

Extraction 56.5 per cent

Assuming that 12.5 per cent more will be obtained from pillars, the extraction will amount to about 63 per cent. At another operation where the average thickness is 5 feet and 316.6 acres have been worked over, the coal available was 2,758,219 tons

Tons produced 1,847,582

Extraction 67 per cent

Assuming a probable extraction of pillars, the final recovery will amount to about 70 per cent. At a third mine, where conditions were favorable because of level coal and the absence of gas, 25-foot rooms were driven on 75-foot centers; the pillar was drawn back half way to the entry as soon as a room was finished, the entry pillars and room stumps being extracted when the entry was abandoned. Under these conditions the recovery was about 85 per cent.

C. H. Nesbitt, Chief Mine Inspector of Alabama, estimates the average recovery to be 80 per cent, the highest percentages being reached in the Pratt and Montavallo beds.† There has been great improvement in the percentage of extraction in the past twenty years, and even in the past ten years. This improvement has been due largely to more nearly complete and accurate mapping, and to more improved and effective methods of controlling the water in slope and shaft mines.

30. *Indiana*.—Almost no information has been available concerning the percentage of coal extracted in Indiana mines. W. M. Zeller† reports that the extraction in the Brazil district is probably about 60 per cent. This estimate agrees fairly well with the estimates of operators in southern Illinois, and since such estimates have been found to be too high in almost all instances, it is probable that the

* Personal Communication.

† Personal Communication.

average extraction in Indiana, as in southern Illinois, will not exceed 50 per cent.

31. *Michigan*.—The coal beds in Michigan are irregular in extent and decrease in thickness with depth. Sometimes they are entirely cut out by erosion or replaced by sandstone and other materials. Usually the beds above the coal consist of black shale, and they are often weak. Owing to erosion, coal is sometimes found directly below clay, sand, or gravel, or below other unconsolidated rocks, where it is practically unworkable. At several mines the roof is of black bituminous limestone. In most instances the floor is of fire clay or shale, although sandstone is sometimes found. The thickness of coal varies from 2 feet, 6 inches to 3 feet, 10 inches, a fair average at Saginaw being 3 feet. In the Saginaw Valley the surface is level.*

R. M. Randall states † that the first company in the district operated within the city limits of Saginaw, and that because of the necessity of leaving pillars to protect the surface the recovery was only about 68 to 70 per cent. At present this company is operating in farming districts where it is not necessary to maintain the surface; and the recovery, within the last five years, has been about 90 per cent. The room-and-pillar system is used with rooms projected 40-feet wide on 50-foot centers and driven 150 feet, but the actual dimensions vary according to the conditions of the roof. Short-wall machines are used for undercutting. It is estimated that 75 per cent of the room pillars and 95 per cent of the entry pillars are recovered, and that the extraction on the advance is 70 per cent. The conditions at the old and at the new mines have been so different that it is impossible to give an average value for the extraction, but it is believed that the extraction in the new mines in the area actually worked will be from 85 to 90 per cent.

32. *Iowa*.—The physical conditions in the Iowa coal field are not uniform. The cover ranges in thickness from a few inches to 300 feet, and consists of the coal measure beds and glacial drift, the latter commonly constituting the larger part of the thickness. The workable coal beds generally have a top of draw shale varying

* Lane, A. C., Mich. Geol. Sur., Vol. 8; Mines and Minerals, Vol. 23, p. 148.

† Personal Communication.

from a few inches to two feet in thickness. Above this is a black shale, or sometimes a bituminous or argillaceous limestone. In the latter case this rock is often strong enough to permit a reduction in the amount of timber used and thereby to facilitate mining. This condition makes it possible in Appanoose County to work a bed in which the thickness of clean coal is only about 25 to 27 inches. The bottom everywhere consists of plastic fire clay, and in the Appanoose field the undercutting is done in this clay. Its occurrence is frequently the cause of creep in room-and-pillar mines. The most unfavorable conditions are found through the northern end of the Iowa coal fields.

Even in single districts the percentage of extraction varies between wide limits. The maximum extraction, estimated at 90 per cent or more, is reached in the Appanoose field, where the long-wall system is employed. An operator* familiar with conditions in these long-wall mines says that the extraction is complete, but that it is less than the previously calculated amount of coal in the ground because of the presence of faults.

In the room-and-pillar districts the extraction rarely if ever exceeds 75 per cent, and under especially bad conditions of bottom and top with an abundance of water, it may not exceed 50 per cent. Probably a fair average of recovery for the state is 70 per cent. The percentages given refer only to the bed mined and to the area of actual mining operations. When larger areas are considered, the percentage of recovery is less because of the loss of considerable quantities of coal through lack of coöperation between owners, a loss estimated to be at least 10 per cent.†

The engineer‡ of one of the operating companies says that in the room-and-pillar mines with which he is familiar the recovery will average about 75 per cent, and that a recovery of 80 per cent is expected in the newer mines.

33. *Missouri*.—The coal fields of Missouri may be roughly divided into three districts, the first district lying near the middle of the state in Macon and Randolph Counties where operations are conducted on the room-and-pillar method, the second district farther

* Taylor, H. N., Personal Communication.

† Beyer, Professor S. W., Personal Communication.

‡ Jorgensen, F. F., Personal Communication.

west along the Missouri River in the vicinity of Lexington where operations are conducted on the long-wall system, and the third district in the southwestern part of the state, where the conditions are similar to those of southeastern Kansas and northeastern Oklahoma and where the room-and-pillar system has generally been followed. Recently a considerable quantity of coal has been obtained in the southwest district by stripping.

In Randolph and Macon Counties, in the neighborhood of Bevier, the coal is considerably broken by faults and horsebacks, and recovery does not exceed 50 per cent.* In the long-wall district the extraction in the area worked out is practically complete, but most of the operations are conducted on a small scale and no estimates covering the probable extraction over the whole area are available.

In the southwestern part of the state the continuity of the coal is considerably broken by horsebacks, as in the neighboring parts of Kansas and Oklahoma. Mining methods have not been highly developed, and no great attention has been paid to completeness of extraction. It is not probable that the extraction in this district, within the areas worked, will be more than 50 per cent.

34. *Arkansas*.—Steel says† that the ordinary waste of coal in Arkansas is unusually great even for this country, a fact to be accounted for partly by unfavorable geological conditions. In addition to the wastes common to all coal producing states there are others due to local geological and physical conditions, which Steel considers unusually unfavorable in Arkansas.

There is considerable loss because of irregularities of entries, due to the varying dip of the bed. Entries which are turned from the slope at standard distances measured along the coal seam will have variable and perhaps severe grades if they are driven straight or will be very crooked if they are driven on grade. If the dip increases and the entries are driven on grade, the distance between entries decreases and sometimes the rooms between entries become so short that the entry from which they are turned is discontinued; then rooms from the entry below are driven long enough to take out all the coal, or part of it, which would have been taken out through the intermediate entry. Sometimes the length of rooms necessary to

* Taylor, H. N., Personal Communication.

† Steel, A. A., "Coal Mining in Arkansas," Ark Geol. Sur.

extract all the coal would be so great that part of it is left. Sometimes this is won through the upper entry, and sometimes it is lost.

There are also losses due to irregularities in the coal, and entries are frequently not extended through areas of low coal to get the good coal lying beyond. Areas of thin coal are commonly abandoned. The losses due to thin or poor coal are greater perhaps in Arkansas than in other states, because the dip of the beds makes the driving of entries around these poor areas considerably more expensive, and it is not profitable to take out good coal lying beyond poor coal unless the area of the good coal is large. There is often considerable loss from the abandonment of parts of beds. In many places the different benches of thick beds are separated by thick partings; if a single bench is thick enough to mine, it is worked separately, and sometimes the bench above it or below it is lost. The loss of coal in this form, though not so great as formerly, is probably greater in Arkansas than in any other state, with the possible exception of Colorado. Loss due to the need of protecting the surface is not serious, because the value of the surface is low, and the rough topography insures good drainage.

H. Denman,* an operator familiar with the district, expresses the opinion that the recovery in both Arkansas and Oklahoma does not exceed 50 per cent. In certain portions of a mine the recovery may be as high as 70 per cent, but he believes if the whole area of the mine is considered, the percentage of extraction will not, in any case, exceed 55 per cent. These statements are applicable to both Arkansas and the neighboring Oklahoma district, as the same system of mining is used in both.

The system of mining in the Arkansas-Oklahoma field is practically the same that was used when the field was first opened about forty years ago. There is no systematic attempt at laying out mines with the view of drawing pillars, but the general plan is to get as much coal as possible in the first working and to abandon the remainder. There is one mine in which an attempt is being made to plan the work so as to obtain the pillar coal, but this attempt is so recent that it is impossible to foretell the degree of its success. The widths of rooms and pillars are influenced by the charges for narrow work and for yardage, which are so high that neither narrow rooms nor long break-throughs can be driven. At present the average room

* Personal Communication.

neck is about 10 feet long, and if longer necks could be driven without increased cost, it might be possible to prevent squeezing of the entry and to obtain a considerable amount of coal from the entry pillars, but the yardage cost is so high that this procedure seems unprofitable.

35. *Kansas*.—The coal produced in Kansas comes from three districts, the one in the southeastern corner of the state being by far the most important. The others are the Leavenworth and the Osage districts. The Leavenworth district lies in the northeastern part of the state and may be considered as connected with the district of northwest Missouri, although the strata dip toward the west and the coal is found at greater depths in Kansas than in Missouri. All operations in the Leavenworth district are on the long-wall system, and the extraction, in the areas mined out, is practically complete. Coal in this district ranges from about 19 to about 24 inches in thickness. The depth is about 700 feet. A 3-foot bed lying at a depth of 1,000 feet was found at Atchison about ten years ago and was worked by the long-wall method, but the work was not commercially successful and was abandoned. The Osage district lies to the south of Topeka and is not important commercially. The coal is about 20 inches thick, and is mined entirely by the long-wall method. The extraction is practically complete within the area mined out. This is the thinnest bed of bituminous coal worked in the United States.

In the southeastern district of the state the coal beds lie on the west slope of the Ozark uplift and dip toward the west and northwest. The beds contain numerous horsebacks which interfere with systematic mining. The room-and-pillar method is followed, and little attempt is made to extract pillar coal. Practically all coal in Kansas, except that produced by the long-wall method and by stripping, is shot from the solid, a method which unquestionably leads to the production of small coal, especially where the holes are greatly overcharged as they usually are. The recovery is in the neighborhood of 50 per cent, although it may sometimes be greater in limited areas, because the horsebacks may be made to serve as pillars. H. N. Taylor, in a personal communication, confirms this estimate of extraction. He says that in places a considerable loss is experienced, because the rate for mining low coal is so high as to be considered prohibitive, and even if the rate is paid it is difficult to

get men to work the low coal. A. C. Terrill reports* that the closest estimates of those most familiar with conditions place the recovery at about 50 per cent.

The approaching exhaustion of the shallower mines has necessitated the working of the northern part of the district where the

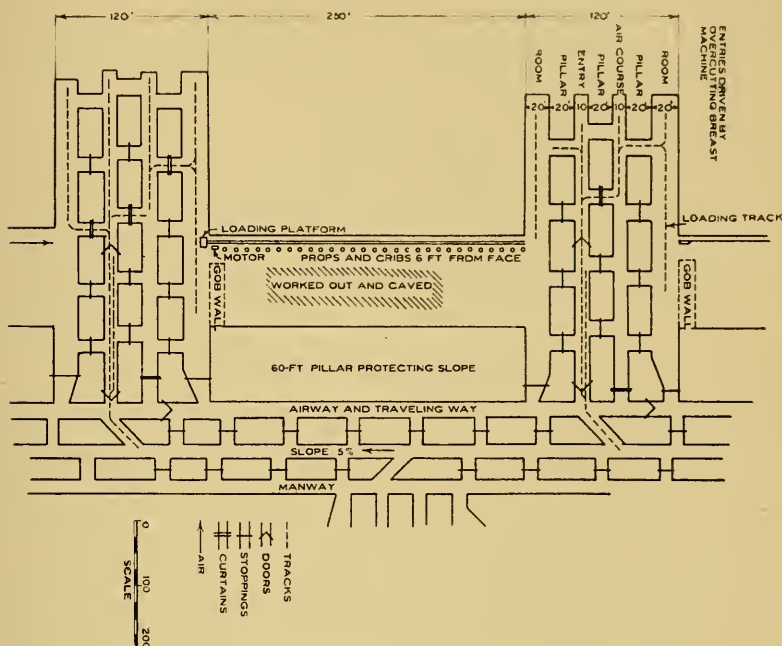


FIG. 38. PANEL LONG-WALL IN OKLAHOMA

cover is about 250 feet thick. At least one of the operators desired to work these deeper mines by the panel method, but it has not as yet been found possible to reach satisfactory arrangements with the mine workers. Since a makeshift adopted to prevent the spread of squeezes leaves two rooms out of seven unworked, the recovery has been reduced about 1,000 tons per acre. The operators still hope that they may be able to introduce the panel system and thus materially increase the recovery.†

In recent years a large amount of coal has been taken from this district and from the neighboring region in Missouri by extensive

* Personal Communication.

† Taylor, H. N., Personal Communication.

stripping operations. In the area worked over, the extraction by this method is practically complete.

36. *Oklahoma*.—In Oklahoma the coal is produced largely by individual operators, the land being owned by the Indian nations and leased to operators in small tracts.* Elaborate plans for mining are not to be expected under these conditions. Elliot estimated that the recovery of the entire bed worked was not more than 55 per cent. He said that the low percentage of recovery was due to the extravagant system of room-and-pillar mining adopted, and that this system could not be changed because of unfavorable labor conditions.

The Rock Island Coal Mining Company obtains an extraction of 48.18 per cent in the McAlester district. In the Hartshorne district this company has five mines, their percentages of extraction being 56.6, 52.6, 55.0, 51.5, and 47.8, respectively. The average percentage of extraction at these five mines is 52.7 and the average of all the mines of the company in Oklahoma is 51.8.†

This company is now trying a panel long-wall plan (Fig. 38) with the hope of increasing the extraction from about 57 to about 70 per cent. The coal is about 3 feet, 4 inches thick, and dips from 5 to 8 degrees. The working face is parallel with the dip. The roof along the face was at first supported by cribs built of 8-inch by 8-inch timbers about 4 feet long and these cribs were withdrawn and moved forward as the face advanced, the roof being allowed to fall. A row of props was also used to support the top above a conveyor used for carrying the coal along the face. At present the use of cribs has been discontinued, except along the ribs of the entries, and 10-inch by 10-inch props are used to support the roof. These are drawn and reset as the face advances. The necessity of using props on both sides of the conveyor constitutes one of the difficulties of the operation. The roof breaks as the face advances. There seems to be no great difficulty in the use of undercutting machines, but sometimes the coal falls too soon for convenience in loading, and large lumps clog the conveyor. While this operation must still be considered in the experimental stage, the working face has been advanced about 130 feet without serious difficulty. It is planned that the pillars flank-

* Elliot, James, "Conditions of the Coal Mining Industry of Oklahoma," Proc. Amer. Min. Cong., p. 221, 1911.

† Scholz, Carl, Personal Communication.

ing the long-wall panel, left during the advance, shall be taken out on the retreat.

37. *Texas*.—There are three bituminous coal fields and one lignite field in Texas. In one bituminous field in the north central portion of the state, practically all the mines are operated on the long-wall plan, and the recovery is nearly complete. The two other bituminous fields are located in the southwestern part of the state along the Rio Grande. In this district the mines are operated on the room-and-pillar plan, and the recovery is said to be about 75 per cent. The lignite field extends entirely across the state from the northeast corner in a southwesterly direction to the Rio Grande. All lignite mines are worked on the room-and-pillar method, and the recovery varies greatly in different parts of the state, but 75 per cent is probably the average.*

38. *North Dakota*.—J. W. Bliss, State Engineer, estimates that the recovery in the coal mines of North Dakota is between 70 and 75 per cent. The manager of one mine claims a recovery of about 85 per cent. †

39. *Colorado*.—The principal producing districts of Colorado are the bituminous district in the southeastern part of the state, near Trinidad, and the lignite district just east of the mountains in the northern part of the state. The bituminous district is the more important. In the Trinidad district‡ the average thickness of the coal is about 6 feet. The top is strong, and the bottom is weak. Entries are driven to a fixed boundary, and the rooms which are needed to supply enough coal to keep the driver busy are turned. When the boundary is reached, rooms are turned at the inby end of the entry, and pillar drawing is commenced as soon as the rooms reach their limits. Nearly all the coal is taken out on the retreat. Rooms have a maximum width of 18 feet, and room pillars are 32 feet wide. All work is done with picks. The coal is soft and occasionally the pillars crush, but most of the difficulty encountered is due to heaving of the bottom. The cover averages more than 600

* Gentry, B. S., State Inspector of Mines, Personal Communication.

† Personal Communication.

‡ Weitzel, E. H., Personal Communication.

feet in thickness. The output from mines in this district is used largely in connection with steel making, and operations are very regular, most of the mines being worked every day in the year. The recovery at a typical mine in this district, calculated for operations over a period of several years, is 87.2 per cent.

In the domestic coal district in the neighborhood of Walsenburg, the average thickness of coal is 5 feet. It is stronger than the coal in the Trinidad district, and a squeeze is very unusual. The cover averages about 400 feet in thickness. As a rule the bottom in this district is stronger than the top, and little difficulty is experienced from heaving. The work is less regular than in the Trinidad district, although it is fairly regular except in March and April when the mines are usually worked about half time. Rooms are driven 25 feet wide on 50-foot centers. There is little difficulty in drawing pillars. The tendency in these districts has been to drive narrow rooms and to leave wide pillars, and this has assisted in increasing the percentage of recovery. The extraction in a typical mine in this district, calculated for operations over a period of several years, is 91.7 per cent. The chief engineer* of another company operating in this same district believes the extraction in certain portions of the mines of his company will reach 80 per cent. In the Canyon district the long-wall system is used, and the recovery is nearly complete.

40. *New Mexico*.—No information is available concerning the percentages of extraction in New Mexico.

41. *Utah*.—The principal coal fields of Utah are located in Carbon County.† The main coal horizon has from two to four workable beds, from 5 to 28 feet in thickness. The main workable bed, known as the Castle Gate, varies in thickness from 5 to 20 feet, and rests on a massive close-grained sandstone. The problem presented by these deposits is one of mining thick seams, comparatively level or slightly inclined. Formerly some seams 4½ to 8 feet in thickness were worked, but at present most of the mining is done in seams varying from 8 to 28 feet in thickness. The physical features to be taken into consideration in this district are: the number of workable seams, the thickness of seams and their relation to one another,

* Personal Communication.

† Watts, A. C., "Coal Mining Methods in Utah," *Coal Age*, Vol. 10, p. 214 and p. 258; and Personal Communication.

the character of the coal, the dip of seams, the character of roof and floor, cover, faults, dikes, wants, the flow of water, sometimes gas, and the burned-out coal beds in place with their residual heat.

Throughout most of the fields there are at least two workable seams, and these are generally found in one coal horizon. In several sections, however, there are three and sometimes four workable seams 5 feet or more in thickness. The distances between these seams vary considerably, so that in some sections there are no unusual problems involved while in others two or more workable seams are found with so little intervening strata that the problem of successful extraction has not yet been solved. There may be, for example, an 8- to 14-foot seam underlying a 6- to 10-foot seam with about 200 feet of intervening strata; in another instance a 5- to 8-foot bed lies 60 feet below one 5 to 11 feet thick, and this lies from 12 to 20 feet below a 22-foot seam, which in turn lies 30 feet below a 6-foot seam. In still another instance an 11-foot bed is found from 3 to 40 feet below a 6-foot bed. There is considerable variation in the physical characteristics of the beds, some being hard and brittle and others tough. In some instances the cleavage is good, while in others it is not pronounced. Almost without exception the coals are hard to cut, and some are hard to shoot. The average dip does not exceed 10 per cent, and in some places the beds are practically flat. As a rule the floor consists of hard smooth sandstone from which the coal parts rather readily. In many cases the roof is of shale varying in thickness from a few inches to several feet. Where a sandstone roof is found, it is generally too hard to break for easy mining. In some places the cover is more than 2,000 feet thick, and there are only a few localities in which it is less than 1,000 feet in thickness. This heavy cover makes the mining of these flat thick seams a serious problem in itself, but the additional complication of great irregularity in depth and the unyielding qualities of the thick beds of overlying sandstone make the problem still more serious. A condition which modifies, at least locally, the laying out and working of a mine is the fact that near the outcrop there are sometimes found large areas of burned coal. These sometimes extend 2,500 feet in from the outcrop. Mining in burned areas is often dangerous, if the burning has been at the top, because of the disintegration of the roof.

With one exception all the mines of the district are opened from the outcrop by means of slopes of drifts. Where conditions of topog-

raphy and property permit, main slopes are driven directly on the pitch of the seams. All mining is by the room-and-pillar method. An attempt to use the long-wall method in one case failed because of the unyielding nature of the roof. The double-entry system is almost universal, although in one case a triple entry is used, and in some cases the double-entry system has been so modified by the connection of the first rooms on the cross entries that it has become practically a 4-entry system. In the earlier workings rooms were

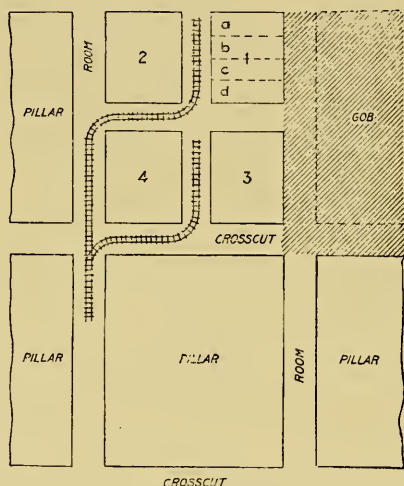


FIG. 39. PILLAR DRAWING IN UTAH

turned from the cross entries as these were driven, but the system resulted in the occurrence of bounces, which seem to take the place of the squeezes that occur with more yielding materials. In later operations the panel system has been used, and the pillars are drawn on the retreat.

Methods of drawing pillars are of particular interest, since they show how almost complete extraction can be attained under conditions which seem unfavorable. These are described by Watts substantially as follows:

In one method (Fig. 39), the block at the end of the pillar on the inby side of the cross-cut is divided by another cross-cut driven through its center, and from the center of this new cross-cut a narrow

room which splits the stump into two parts is driven to the gob. The upper inside stump is taken out first by slices beginning at the inby end; then the remaining stump is removed. The lower half of the original block or pillar meanwhile is split by a narrow road, and the process is thus continued down the pillar, each block being divided into four parts.

Pillar drawing in a flat seam 12 to 14 feet thick under a cover 800 feet thick and under a roof which broke fairly well when posts were removed has been successfully accomplished by the following

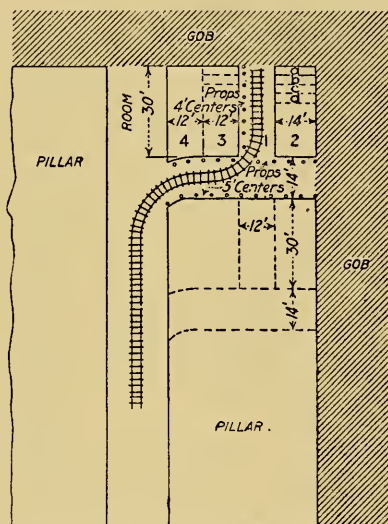


FIG. 40. PILLAR DRAWING IN UTAH

method: 20-foot rooms were driven with 50-foot pillars (Fig. 40), and a cross-cut was driven through the pillar; thus a 30-foot stump was left. This 30- by 50-foot stump was then split by a 12-foot room which left a 24- by 30-foot stump next to the room and a 14- by 30-foot stump on the other side. The latter stump was then taken out in slices which begin at the gob, and the roof was supported by props set every 4 feet. The coal was undercut by hand and shot with black powder. When this block had been removed, the track was taken up, and all props were drawn except a row adjacent to the rib of block No. 3. These blocks were numbered in the order of their extraction, 1, 2, 3, and 4. Block No. 3 was then taken out from the

cross-cut to the gob, track was laid in the space, and block No. 4 was taken out in the reverse direction, that is, beginning at the gob. In this mine 3 to 6 feet of top coal are left up to protect the roof on the advance, but this coal is taken down on the retreat. When cross-cuts are made in pillars preparatory to drawing them the whole height of the seam is taken.

Pillar drawing in 16-foot coal with a cover of 400 to 1,000 feet, with no top seam and with the roof breaking well when props are drawn, is accomplished as follows: Rooms about 400 feet long are

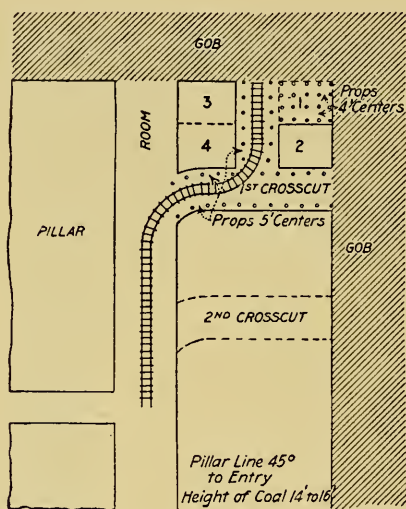


FIG. 41. PILLAR DRAWING IN UTAH

driven straight up the pitch which averages about 10 per cent. Pillars are drawn on the retreat, and the line of break is kept at an angle of 45 degrees with the entry; thus work is done on six or seven pillars at a time. Rooms are about 20 feet wide, and pillars are 50 feet wide. A cross-cut is driven through the pillar 30 to 35 feet from the end (Fig. 41); thus a block about 25 to 30 feet by 50 feet is cut off. This block is then split by a room about 12 feet wide. Blocks 1 and 2 are drawn by slicing which begins at the end next to the gob. The top is supported by means of props at 4-foot intervals, and after the two blocks have been removed and the track has been taken out, these props are pulled, and the area is allowed to cave. The track is then laid in the main room, and blocks 3 and 4 are taken out by

end slicing from the room; then the track is taken out, and the props are pulled. Another cross-cut meanwhile has been made through the pillar nearer the entry, and a room has been driven through the stump so that by the time the first stumps have been extracted, work is being begun on the lower stumps.

In this mine the size and the systematic placing of props have an important bearing on the successful recovery of the pillars. Until heavy pine props were used, trouble was likely to occur at any time. Now props as large as 10 inches in diameter at the small end with

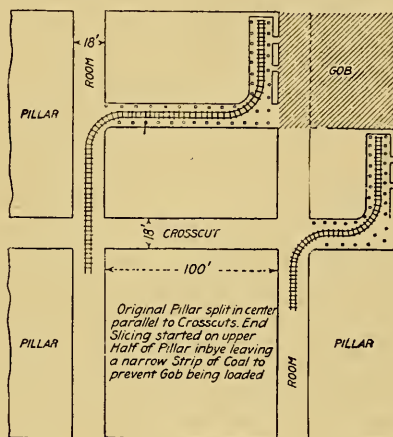


FIG. 42. PILLAR DRAWING IN UTAH

correspondingly heavy caps and, in some places, cross bars are used. Props are set at regular distances. Many of these props are recovered and re-used three or four times. By the adoption of this method, the safety factor is largely increased, the percentage of recovery is greater, and the product is of better quality. In some cases it is customary to mark pillars at regular distances so that the mine foreman or pillar boss may easily determine the progress of the pillar work daily and may keep the ends of the stumps in proper alignment.

Another method of pillar drawing sometimes used is similar to that last mentioned, although the stump left is a little shorter. This stump is then split into quarters, and the work of extraction proceeds from the cross-cut toward the gob, a thin section of coal being

left to the last around the edges of the block to prevent the mixture of fallen roof with clean coal.

A plan which has proved satisfactory, partly because it does not split the pillar into many small stumps, is illustrated by Fig. 42. The original block of pillar coal 100 feet wide by 120 feet long is divided into equal parts by a cross-cut, and the upper half is taken out by slicing beginning at the inner side, a curtain of coal being left to prevent the loading of gob, and one or two rows of props being put along the side of the coal. Before the track and props are pulled, most of this section of coal is loaded out. In this case the rooms are 18 feet wide and the pillars 100 feet wide.

Little information is available regarding the percentage of extraction in Utah. There is probably only one mine in the state which has been worked out, and no reliable information can be obtained concerning this mine. It is believed, however, that the extraction was probably about 75 per cent. In coal 12 to 16 feet thick and under cover varying from 200 to 2,000 feet an extraction as high as 90 per cent has been made, if marketable coal alone is considered. If all the coal in the bed is considered, the recovery is about 80 per cent. In beds ranging from 15 to 30 feet in thickness, retreating work has hardly been started so that no information on total recovery is available. It is possible that it will be rather low. It could be made higher if the filling method could be used, but the price of coal does not warrant the use of this method.

A condition largely influencing the percentage of extraction is the presence of more than one workable seam with little intervening material. Under present conditions the percentage of extraction from an area containing seams with 3 to 12 feet of intervening rock is at best only 65 per cent of all the coal. In one mine an attempt was made to take out the coal from two beds, the lower being 11 feet thick and the upper 5 to 6 feet thick with intervening rock $2\frac{1}{2}$ to 12 feet thick. The workings were in the lower bed, and frequently the roof caved as soon as the pillars were drawn and practically all the upper seam was lost.

A. B. Apperson* gives the percentage of extraction in two mines as nearly 95 per cent of the total seam, while the extraction at another mine is about 85 per cent. At the mines yielding the lower percentage of extraction, the cover is about 800 feet. At one of the mines

* Personal Communication.

yielding the higher percentage of extraction, pillar drawing was commenced at the middle of the mine under a cover of approximately 1,700 feet. A good break is obtained about 50 feet behind the pillar extending the full length of line. Only small areas have been worked out in these mines.

42. *Washington*.—No reliable information is available concerning the percentage of extraction in Washington. Conditions are somewhat unusual in that most of the coal has been badly folded and faulted and consequently crushed, and the deposits have been steeply tilted.* It is impossible to separate the refuse in the mines, and a large percentage of it has to be washed.

* Daniel, Professor Jos., Personal Communication.

APPENDIX

DEVELOPMENT OF MINING METHODS IN ENGLAND
AND ON THE CONTINENT

43. *Brief History of Coal Mining Practice in England.*—It is interesting to review briefly the history of the coal mining methods of England, because the mining methods employed in this country are largely applications of methods developed in England and brought over by miners.

The many methods of obtaining coal may be grouped on the basis of recovery under two main headings: one in which the whole of the coal seam is taken out in the first working, and another in which

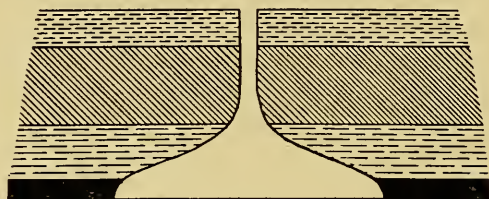


FIG. 43. BELL PIT

only a part of the seam is removed in the first working. These may be called the no-pillar, or long-wall, system and the pillar system.

The earliest mining was naturally done on the outcrop of the seams, and as this practice became difficult or impossible, the use of "bell-pits" (Fig. 43) was developed. These were holes or shafts, from 3 to 4 feet in diameter, which were sunk through the shallow overburden near the outcrop and widened out at the bottom in order to allow the excavation of as much coal as possible without permitting the roof to fall in. It was of course impossible to extract much coal from a pit of this kind, and in order to obtain the coal even from a small area it was necessary to dig a large number of pits. This method was gradually abandoned, and the coal was worked by means of galleries driven out from the bottom of the shaft, usually in an unsystematic manner; thus began the use of pillars to sustain the roof. The driving of galleries permitted the working of much

greater areas than could be reached from the bell-pits; however, no areas of more than a few acres were worked from one shaft, nor were systematic ventilation and regularity in laying out the workings introduced until the exhaustion of the shallow coal made necessary a study of methods to be employed in deeper workings. Until the introduction of the Newcomen engine, when pumping by steam power became possible, shafts were rarely as deep as 200 feet; they

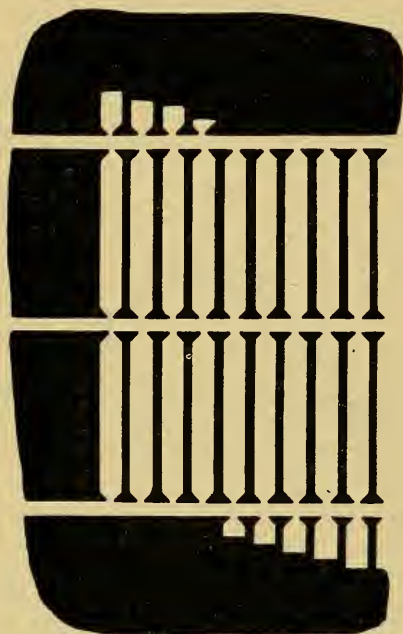


FIG. 44. BORD-AND-PILLAR

were 7 or 8 feet in diameter, and the area worked from one shaft was seldom more than 600 feet in radius.*

The structure of many coal seams is such that there are two directions, determined by the cleat of the coal, in which the seams can be most easily worked. The direction at right angles to the face cleats is known as "bordway," while the other direction approximately at right angles to the first is known as "headway." The excavations made in a direction at right angles to the principal or face cleats

* Bulman and Redmayne, "Colliery Working and Management," p. 3, 1906.

were called bords, and, as the coal was most easily taken out in this direction, these excavations were made wider than the connecting passages or headways. The coal left in place to sustain the roof was called pillars; thus originated the term, "bord-and-pillar" method, which in its various developments is commonly known in this country as the room-and-pillar method. This method was developed in different forms in England and was variously called "bord-and-pillar," "bord-and-wall," "post-and-stall," and "stoop-and-room."

In early times the "pillars" were probably made very small and square measuring from 3 to 6 feet each way. In the eighteenth century the bords were usually made 9 feet wide and the pillars 12 feet

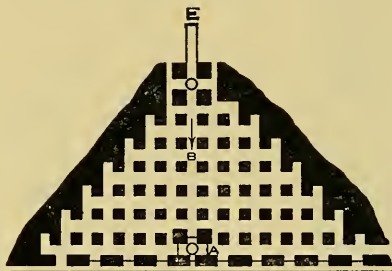


FIG. 45. STOOP-AND-ROOM

wide, though they were of course irregular. The bords were commonly widened out between the headways (Fig. 44), and the pillars were thus gouged to as great an extent as was considered safe, it being desirable, in view of the comparatively small area which could be reached from a single shaft and in view also of the inadequate ventilation, to extract as much coal as possible within the area worked. This method of working was essentially wasteful as not much more than 50 per cent of the coal was obtained, and since the pillars left were unable to bear the weight of the cover, they were soon crushed and further working was made impossible.* Possibly a larger percentage of coal was taken out in some places as Redmayne † says it was rare that more than 65 per cent of the available area could be

* Boulton, W. S., "Practical Coal Mining," Vol. 1, p. 296.

† Redmayne, R. A. S., "Modern Practice in Mining," Vol. 3, p. 82.

extracted; but he refers to John Buddle as saying, at a somewhat later period when the pillars were 18 by 66 feet, that not more than $45\frac{1}{2}$ per cent of the contents of a fiery seam could be obtained under any method of working then known. It was not until much later that the exhaustion of the most easily worked deposits directed attention to the desirability of higher extraction.

A system with square pillars and working places of almost uniform width (Fig. 45) has continued in common use at Whitehaven and in Scotland down to the present time. In the north of England the pillars were usually oblong, probably because the highly developed face cleat of the coal made the extraction in one direction much easier than in others.* The lengthening of the pillars reached its greatest extent in South Wales where cross holing was so little employed as scarcely to form a part of the system of working.

The date at which the extraction of pillar coal was begun is not known, but it seems certain that pillars were removed in the north of England before 1740. The following statement† is made concerning the removal of pillars:— "The documentary evidence cited goes to show that, previous to 1708, the general practice was to leave small pillars of coal standing for the support of the roof; 30 years later pillars were being partially, sometimes entirely, removed; and during the remainder of that century, in mines free from gas, a second working of the pillars was frequently carried out. In the deeper and fiery collieries, which began to be developed about the middle of the eighteenth century, the risk of creep as well as of gas explosions prevented the removal of the pillars. The invention of the safety lamp, improvements in ventilation, and the formation of much larger pillars in the first working . . . were introduced during the first 30 to 40 years of the present (nineteenth) century . . . which enabled the pillars to be removed in a second working."

Concerning the size of pillars, Jars, a French engineer who published "*Voyages Métallurgiques*" in 1774, says in "*A Journey Through the North of England*," that underground pillars of coal were made from 39 to 54 feet square, and that working places were from 5 to 16 feet wide. At this time the pillars were left until all the coal was exhausted. Another traveler who made a tour of Scotland in

* Galloway, R. L., "*Annals of Coal Mining and the Coal Trade*," p. 181, 1898.

† Bulman and Redmayne, "*Colliery Working and Management*," p. 14, 1906.

1772 said that pillars 45 feet square were left and that not more than one-third of the coal was worked.*

The extraction of a portion of the pillars in gassy mines by a second working was just beginning to be a regular part of the bord-and-pillar system at this period. It could, however, be effected only in a very incomplete manner so long as the miners had to depend upon candles and steel mills for light. At this time also the extensive adoption of the long-wall system began.†

In the early part of the nineteenth century little change seems to have been made in the size of pillars used in the Newcastle district, according to a statement of an author who speaks of them as being 60 by 27 feet or, in some instances, 27 feet square. About this time the drawing of pillars seems to have become common in Northumberland, as Mackenzie, who wrote a "View of Northumberland" in 1825, speaks of the mode of working coal as being much improved in the last few years. He says (second edition, page 90), "from seven-eighths to nine-tenths of the coal is at present raised, whilst formerly but one-half, and frequently less, was all that could be obtained." No doubt this statement refers to the general practice of removing pillars, which had been made practicable in gassy mines by the invention of the Davy lamp.

Conflicts of interests between coal producers and owners of the surface are of early record. It was, of course, the desire of the colliers to remove as much of the coal as possible, even where the surface was supposed to be maintained, and the result of making pillars too small was subsidence. There is probably no definite record of the first occurrence of subsidence, but one of the earliest mining leases written in the English language, dated 1447, indicates that it was the custom to leave pillars to sustain the surface and that subsidence had already taken place.‡

In the latter part of the eighteenth century the working of pillars in a fiery mine, such as Wallsend Colliery, was not considered practicable, and only about 39 per cent of the coal was obtained while 61 per cent was permanently lost. This coal was at a depth of 600 feet, and the workings represent the best practice of the bord-and-pillar system at that period.§

* Galloway, R. L., "Annals of Coal Mining and the Coal Trade," p. 353, 1898.

† Ibid, p. 362.

‡ Ibid, p. 69.

§ Ibid, p. 293.

Until about the end of the eighteenth century an extraction of 45.5 per cent was considered the maximum which could be obtained in the deep collieries of the Tyne.* The first person to offer a partial remedy for this very unsatisfactory condition was Thomas Barnes, viewer of Walker Colliery, who projected a scheme in 1795 for recovering a portion of the pillars without causing loss of the mine. This system provided for dividing the workings into small sections of 10 to 20 acres and isolating these with artificial barriers formed by filling the excavated spaces with stones and refuse for a breadth of 120 to 150 feet. By this method one-half of alternate pillars, or one-quarter of the remaining coal, was removed, and the percentage of extraction was increased from about 39 to about 54 per cent. Wherever pillars were thus removed, a squeeze was brought on, but the barriers kept it from spreading. This method proved to be successful, and it was adopted at other collieries.

Probably about this time the pillars at Wallsend Colliery were left larger as a preparatory step toward a second working. Buddle said that after about one-third of the colliery had been worked by means of 36-foot winnings (12 feet to the bord, 24 feet to the wall or pillar) in which no more coal was left in pillars than was considered sufficient to support the roof, the size of the winnings was increased to 45 feet (15 feet to the bord and 30 feet to the pillar). "This change of size," he said, "was not made for the purpose of obtaining a greater produce in the first working of the seam. But the notion of the future working of the pillars then began to be entertained, and the increased size of the winnings was considered a more favorable apportionment of the excavation and pillar for the attainment of this object." This is the first record found of a second working in the deep Tyne Collieries.† Pillars seem to have been worked in the northern part of England about the middle of the eighteenth century.

44. *Ventilation*.—The distance to which workings could be driven and the extent to which pillars could be drawn, especially in gassy mines, were found to depend largely upon ventilation. In the latter half of the eighteenth century improvements in ventilation, which had been used earlier in the Cumberland field, were introduced into

* "Trans. Nat. Hist. Soc. of Northumberland," Vol. 2, p. 323.

† Galloway, R. L., "Annals of Coal Mining and the Coal Trade," pp. 315-318, 1898.

the north of England, where the frequency of explosions made better ventilation necessary. Until this time it had been considered sufficient to conduct the air current along the working face, an arrangement known as "face-airing;" consequently, the worked-out places, which were behind the miners in advancing work, were left without ventilation. As long as the extent of workings was very limited, this method was not attended with great danger; but, as the mines became deeper and more gassy, workings were made larger and the danger from this inadequate ventilation increased, because the worked-out places became magazines for the accumulation of fire damp.

The improved method of ventilation, which was known as "coursing the air," consisted in so directing the air that the whole current passed through all the openings in the mine. While this method was effective in preventing the accumulation of standing gas, it introduced a great danger in that the air took up constantly increasing quantities of gas in its passage through the mine, and, since it was constantly exposed to the lights of miners, it became dangerous in the latter part of its course. It was, moreover, constantly contaminated by the breathing of men and animals and by the smoke from the candles. This method was introduced in the north of England about 1765 or 1766, and it was about this date that the steel mill also was introduced for the purpose of giving light.* Though this method was fairly satisfactory in small mines, it was very unsatisfactory in large ones. At Walker Colliery, although the pits were only half a mile apart, the air current traversed a line exceeding thirty miles in length. At Hebburn Colliery the air course was also said to be not less than the same length. Not only was it difficult to keep the air passages open and the doors and stoppings tight, but the friction of the air limited the velocity of the ventilating current, which would have been low at best since the force causing this current was supplied only by a furnace. At this colliery the circulation of five or six thousand cubic feet of air per minute was considered sufficient, and the velocity was about three feet per second.

45. *The Panel System.*—There was great difficulty in carrying on work in the deep collieries of the North, because squeezes occurred. A method of working described as common in the North at this period consisted in having bords 12 feet wide and 24 feet apart

*Galloway, R. L., "Annals of Coal Mining and the Coal Trade," p. 279, 1898.

connected by headways 60 feet apart, thus leaving pillars of coal 24 by 60 feet. Another size of pillar given is 30 by 72 feet.*

Early in the nineteenth century, John Buddle, Jr., who had succeeded his father as manager at Wallsend and who was responsible for important improvements in coal mining methods, devised and put into practice improved methods of working and ventilation whereby squeezes were effectually kept in check, and the existing system of ventilation was greatly improved. He effected these improvements by dividing the workings into independent districts or panels, as Barnes had done. Buddle's idea, however, was to provide for confining the movement by separating the districts or panels with barriers of solid coal left in the first working. This method was adopted in developing the Wallsend G pit in 1810. Buddle's improvement in ventilation involved dividing or splitting the current. This method of ventilation proved successful and was quickly adopted at other mines to which it could be applied, but the air currents employed were still very feeble.

From the preceding descriptions it will be seen that all the essentials of the room-and-pillar system as now practiced in this country had been developed in Great Britain prior to 1810.

46. *Square Work of South Staffordshire.*—In the Thick seam of South Staffordshire where the coal varies in thickness from 18 to 36 feet, a method which bears a close resemblance to the panel method was developed. The district had been greatly troubled with fires due to spontaneous combustion, and in order to extinguish these fires easily or to confine them within the immediate vicinity of their origin this method, known as "square work," was developed. It consists in dividing the area to be worked into a number of large chambers termed "sides-of-work," surrounded on all sides by panels of solid coal known as "fire ribs." The only openings in these panels are those necessary for the extraction of coal and for ventilation. The panels are nearly square, and from four to sixteen pillars, the number varying according to the size of the chamber, are left to support the roof. Fig. 46 shows an old form of square work. Under the system in its simple form and in the first working, only from 40 to 50 per cent of the available coal is recovered, but the larger portion of that left is recovered by second or even third

* *Ibid.*, p. 395.

workings carried out after the lapse of some years. The final loss in working may, therefore, not exceed 10 per cent of the available coal; the coal recovered in these later workings, however, is frequently badly crushed.*

47. *The Long-wall System.*—The other general method of coal mining, the long-wall method, has been from early times prevalent in Shropshire, from which district it has spread into others. The date of the origin of this method is doubtful, but it is said to have been in general use in the Shropshire district about the middle of the nineteenth century.†



FIG. 46. OLD SQUARE WORK

The long-wall method of mining has been highly developed in England and Scotland and has been applied at greater depths and to thicker beds of coal than it has been in this country. Considered from the point of view of completeness of extraction, the system fulfills the highest requirements: It not only permits, but requires the excavation of the whole bed of coal. Whether all the coal shall be taken out of the mine depends of course on whether it is marketable.

This method of working has not been as generally applied in the United States as have the various forms of the room-and-pillar system. There are, however, certain districts in which it is used almost exclusively. Among the most prominent of these is the long-wall district of Illinois which has been described as District I in Chapter II. The other districts in which the long-wall method is used are those of northwest Missouri, northeast Kansas, the

* Redmayne, R. A. S., "Modern Practice in Mining," Vol. 3, p. 116.

† "Trans. North of Eng. Inst. Min. Engrs.," Vol. 2, p. 261.

Osage district of Kansas, the Appanoose district of Iowa, the north-central district of Texas, and the Canyon district of Colorado. Scattering applications of the method are found elsewhere, but the physical conditions in the regions mentioned have been best suited to its use. It seems probable that the long-wall system, with modifications perhaps, will be more widely used in this country in the future.

48. *Percentage of Recovery in England.*—The methods followed in England have not been developed with the purpose of obtaining a high percentage of recovery. It was not until 1854* that a special department for collecting and publishing mineral statistics was created, and not until 1861 that any systematic estimate of coal resources was made. About this time predictions forecasting the exhaustion of the coal supply within a century caused a great disturbance, and a royal commission was appointed in 1866. A report of this commission was made public in 1871.

The part of this report which deals with waste in working is of special interest. The commission estimated the "ordinary and unavoidable loss" to be about 10 per cent, though they said, "In a large number of instances, when the system of working practiced is not suited to the peculiarities of the seams, the ordinary waste and loss amount to sometimes as much as 40 per cent." The principal part of this unavoidable waste arises from the crushing of pillars.

In addition to this unavoidable loss, there is waste or loss, variable in amount, but sometimes very great, arising from the following causes:†

(1) The leaving below ground or consuming in large heaps of small coal on the surface (presumably the loss from this source is much less at present because of the greater consumption of small sized coal, as in this country).

(2) Undercutting, often wastefully made, in good coal.

(3) The leaving, either wholly or in part, of an adjoining or neighboring bed when it becomes crushed and unworkable, because it is not wanted at the time, or because if it should be worked, the cost per ton of the coal extracted is increased.

* Digest of the Evidence given before the Royal Commission on Coal Supplies, Vol. I., p. IX., 1905.

† Ibid, p. XXXIII.

- (4) Existence of coal on properties which are too small to be worked alone or in small parts of collieries cut off by a large fault.
- (5) Disputes over the cost of drainage.
- (6) The breaking in of water from the sea or from a river estuary.
- (7) The leaving of barriers around small properties or crooked boundaries.
- (8) Lack of plans or records showing the extent of old workings, operations of seams not sufficiently proved to justify expenditure for sinking pits; sufficient information might have been obtained if records of previous explorations had been preserved in available form.
- (9) The piercing of water-bearing strata by shafts and bore holes which are not protected by water-tight casings, or are not carefully filled and puddled when temporarily left or abandoned.
- (10) The cutting through of main faults serving as natural barriers to keep back water and the consequent flooding of the coal.
- (11) The leaving of large areas of coal in populous and manufacturing districts to support the surface and the buildings.

While some of the causes mentioned do not apply directly to conditions in this country, the list furnishes a complete synopsis of reasons for coal losses.

Since the issuance of the report of 1871, there have been great improvements in the methods of getting coal. At the present time the long-wall system is in general use, and the waste has been lowered; yet in some parts of the United Kingdom, notably Northumberland, the pillar-and-stall system is still in general use.

Among the factors contributing to a higher rate of recovery is the greatly increased value of small sizes of coal. It was computed in 1871 that the average value of the small coal mined in Great Britain was only 60 cents per ton, while in 1905 the small sizes of steam coal from the South Wales district brought about \$1.90 per ton; in all the other coal fields the value has been doubled and even trebled. The principal cause of this change lies in the improved preparation of coal. The manufacture of producer gas on a large scale and the growth of the briquet industry have also increased the possible uses of the small sizes. One of the effects of the increase in the value of small coal has been some decrease of the comparative advan-

tage of the long-wall system, since the production of a large amount of fine coal with the pillar-and-stall system is less objectionable than formerly.*

Interest in the subject continued, and another investigation, more exhaustive than the earlier one, was made by the Royal Commission on Coal Supplies which organized in 1902 and presented its report in 1905. The Royal Commission of 1905 adhered to the limit of depth, namely 4,000 feet, established by the earlier commission. It was thought that, although there might be no insuperable physical or mechanical difficulties in the working of beds at greater depths, the expense would be so great that imported coal could be obtained more cheaply.

With regard to thickness, the commission which reported in 1871 had included seams exceeding one foot in thickness as workable. The question is largely a commercial one, and thinner seams are being worked now than formerly. Mr. Gerrard, inspector of mines for the Manchester district, obtained from all the inspection districts returns which showed that in 1900 17.7 per cent of the entire output was taken from seams not exceeding three feet in thickness.† In the United States, limits of 3,000 feet in depth and of 14 inches in thickness have been decided upon by the Department of the Interior as factors determining what portions of the remaining public lands shall be considered coal lands.‡

The Royal Commission took evidence also on the cost of working, and gave figures which show how greatly the labor cost rises and the individual output declines as thinner beds are mined. Mr. Gerrard gave the underground wages as ranging from \$1.68 to \$2.28 per ton in seams up to 12 inches, and from 63 cents to \$1.36 in all underground seams in his district from 1 foot, 1 inch to 3 feet, while the daily output ranged from one-half ton to 3¼ tons. It was estimated that the cost of digging, loading, and hauling in Scotland was \$1.24 for a seam 14 to 15 inches thick, and 65 cents for one from 2 to 2½ feet thick, while the daily output varied from 22 hundred-weight to 1½ tons. In Somersetshire the average cost of working thin seams has been about \$1.92 per ton for a number of years, while

*Digest of the Evidence given before the Royal Commission on Coal Supplies, Vol. I., p. XXV., 1905.

†Digest of the Evidence given before the Royal Commission on Coal Supplies, Vol. I., p. XXXV., 1905.

‡Fisher, Cassius A., "Standards Adopted for Coal Lands of the Public Domain," U. S. Geol. Sur., Bul. 424, Ashley and Fisher, p. 63, 1910.

in Yorkshire the cost in 1900 varied from about 96 cents to \$1.68.* The Commission of 1905 finally decided to retain the figure of one foot as the limit of thickness.

In connection with the subjects of depth and thickness, it should be noted that it is not the practice in Europe to work single thin beds at great depths. The thin beds are worked in conjunction with thicker ones, and it is the lower cost of production in the latter which makes the working of the thin ones commercially possible. The high cost of working thin beds is partly responsible for the high cost of European coal. The American practice is distinctly different, for there are few, if any, districts in this country in which any bed of bituminous coal is worked unless it is believed that such working shows a profit without reference to other workings. Instances of the working of more than one bed of bituminous coal from the same shaft are rare in the United States.

49. *Percentage of Coal Lost.*—A detailed inquiry was made by the Royal Commission into the various sources of loss. The points of greatest interest in connection with the present study were covered as follows:†

“Coal left for Support.—It is evident, that, except in very special cases, it is not possible to remove all the coal. A certain amount must be left in order to maintain shafts, etc., and to support the surface—as, for instance, under houses, railway, canals and rivers—and there seems little hope under existing circumstances of avoiding this source of loss. The amount of coal left for support depends largely upon whether its value is greater than the damage, which would be caused by its removal. . . .

“Barriers.—We have evidence that much coal has been and is lost through the practice of leaving unnecessary barriers between royalties and properties; but the present tendency to take large areas under lease is reducing the loss from this cause, and in many cases barriers between properties are now worked out by mutual arrangements.

“Thick Seams.—Where the seams are of abnormal thickness much coal is, in some cases, wasted, and for various reasons. Sometimes it

* Digest of Evidence given before the Royal Commission on Coal Supplies, Vol. I., p. 178 et seq., 1905.

† Digest of the Evidence given before the Royal Commission on Coal Supplies, Vol. I., p. XXXVI., 1905.

is considered that the whole seam cannot be taken out with safety, and part is therefore left to form a roof. Further, such thick seams are more difficult to work, and when the whole of the seam is not of the same quality, there is a temptation to take out the best coal first and to leave the rest for possible future working. Suggestions have been made by some of the witnesses as to the best method of working such thick seams, and there is little doubt that improved methods combined with the increasing use of inferior coal will to a large extent obviate the difficulties mentioned.

"Inferior and Small Coal Left in Mines.—According to the evidence inferior coal is frequently left in the mine owing to its being unsalable, and in some districts considerable quantities of small coal are also left. In recent years there have been vast improvements in the methods of, and the appliances for, preparing and utilizing small and inferior coal, and the higher appreciation of such coal should go far to put an end to this waste."

Table 10 presents the conclusions of the commissioners of different districts regarding the deductions which should be made to cover losses in calculating the amount of coal remaining available.* It is to be understood that the values given do not refer merely to the losses within a definite mined-out area but to the total losses which are to be expected in extracting the total coal remaining available. Since both amounts of losses and reasons for them are governed largely by local conditions, it is unnecessary to go into details, especially since it was found impossible there, as it has been here, to arrive at definite statement for the losses in all cases. Values, however, are founded upon the opinions of men familiar with the practice in the districts, and they are at least approximately correct.

TABLE 10
PERCENTAGES OF COAL LOSSES AS ESTIMATED BY THE
ROYAL COMMISSION OF 1905

District	Per Cent lost	District	Per Cent lost	District	Per Cent lost
South Wales and Monmouthshire.....	20.68	Warwickshire	2.20	Northumberland	22.61
Forest of Dean.....	15.50	Leicestershire	26.00	Durham	20.23
North Staffordshire	17.00	Lancashire	26.70	Cumberland	28.20
Shropshire.....	13.00	Cheshire	18.70	Scotland	26.20
South Staffordshire.....	27.50	Flintshire	27.60		
		Denbighshire	33.30	Average	21.28

* Ibid, p. XXV.

In the introduction to the report of the Commission, prepared by the editor of the "Colliery Guardian," it is stated (p. xxvi), "Much of the evidence goes to show that the more general adoption of the long-wall system in recent years has resulted in an increased yield of coal. But there are many localities where the conditions are not considered favorable to long-wall working, and where pillars are still left. In the worst cases, in exceptionally bad ground, as much as 50 per cent of the coal is often left behind for this purpose, only to be crushed and oxidized and rendered unfit for future recovery. In the under-sea workings in Cumberland as much as 75 per cent is thus left behind. Perhaps the most interesting point brought out in the evidence is that which concerns thick seams. It certainly does seem unfortunate that where there are 9 feet of good coal in a single seam, nearly one-third of this should be left behind. Yet this happens in many of our thickest seams, and the loss threatens to be still more serious as the depth increases."

50. *Mining Conditions on the Continent.*—In the Franco-Belgian basin the beds are for the most part thin, and they are worked, to a considerable extent, at greater depths than those reached in the United States. In Westphalia the beds are mostly steeply dipping, and in Upper Silesia there are combinations of steep dip with great thickness of coal. The development of mining methods in the United States up to the present time has not been affected by practice in these districts.

51. *Percentage of Extraction on the Continent.*—In France it is the custom to extract as much coal as possible from the bed and to fill the resulting space with rock or other material. The filling material is usually transported to its destination in cars, and the method of packing depends largely on the inclination of the bed. In steeply dipping beds the material is allowed to run into place by gravity, but where the slope is not sufficient to permit this method of packing, it is packed by hand. This custom does not entirely prevent subsidence, but it permits the extraction of nearly all the coal without serious disturbance of the surface. While the method of packing followed in these districts permits the removal of nearly all the coal, the removal is accomplished at an expense which would be regarded as prohibitive in the United States in view of the narrow margin between

cost of production and selling price here. The method of filling by flushing is coming into use in France, but has not yet displaced dry filling in most of the mines. Whatever system of filling is used, and whether the coal is taken out by pillar or long-wall method, the extraction is nearly complete.

Some of the most difficult problems found in any coal mining district have been encountered in Belgium. There is no other country in which such thin seams are worked and in which coal is generally mined at such great depths. At Quaregnon a series of thirty-three seams is worked, the average useful thickness being 1 foot, $3\frac{1}{2}$ inches, while the greatest thickness is 2 feet, 2 inches. These beds vary in dip from 8 to 90 degrees. The flatter portions of the bed are worked by long-wall, and the steeper parts by inverted steps forming an interrupted long-wall face. Other beds of nearly the same thickness are being worked, and it appears in all cases that those thin beds are attacked by some form of long-wall working in which, of course, the extraction is practically complete.* The discussion of these districts is much briefer than their importance as coal mining districts would warrant were it not for the fact that the methods used would not in general be adaptable to physical and commercial conditions in this country. They furnish interesting illustrations of high percentages of extraction under difficult conditions, but can hardly be regarded as indicative of what it would be possible to do in the United States.

In the Westphalian district in Germany large amounts of coal have been lost, not so much as the result of poor mining methods or lack of attention to completeness of extraction as because of the necessity of preventing subsidence of the surface. This region is one of great industrial activity, and surface values have so increased within the last half century that high extraction without filling has become impossible. At first, hand filling was employed, the material used being the waste produced in the large amount of rock excavation necessary in beds lying at various angles combined with slack from collieries where coke was not made. More recently the method of hydraulic filling has been introduced. Where the packing is well done and the mining conditions are favorable, the loss of coal is possibly not more than five per cent, which may be considered a fair estimate of the

* Digest of the Evidence given before the Royal Commission on Coal Supplies, Vol. I., pp. 41, 76, 393, 1905.

loss even where the long-wall method is followed. There is, however, a greater loss in some of the thicker steeply dipping beds, though it has not been possible to obtain estimates of the amount.

The Upper Silesian coal field,* situated in the southeast corner of Prussia and extending into Austria and into Russian Poland, has an area of 2,160 square miles. The character of the seam varies considerably both in composition and in thickness, and thick seams occur only in the northern portion of the field where they are very numerous and many of which are of great thickness.

In this coal field, the problem of removing coal beds of great aggregate and individual thicknesses without serious disturbance of the surface has been met by the development of sand flushing processes of filling. This method of filling was borrowed from the anthracite district of the United States where it had first been used.

The mines are worked with and without sand filling. In the method without sand filling much coal is left unworked in the form of pillars and as support under towns or villages. There is a considerable loss resulting from the difficulty of extracting coal left as barriers between the working places and in the old workings. There is also a considerable loss because of fire. The estimated total loss under this method is 25 per cent.

At present sand filling is being used more or less extensively in most of the mines in the thick beds. It is especially advantageous where spontaneous combustion is prevalent and where surface support is necessary. With sand filling when only a part of the coal is replaced by sand it is estimated that the loss of coal is 10 to 15 per cent; with complete replacement of coal by sand filling, the loss is only from 3 to 5 per cent. Smaller and cheaper timber is used in this case, and the greater portion of this timber is recovered for future use. In four mines in Upper Silesia in which sand filling is used extensively and in sufficient quantities to suit the conditions of the mines, the cost in the seams is between 12 and 18 cents per ton. The cost is variable, however, and is calculated in different ways. The average working cost per ton of coal at the surface in this district is \$1.51, of which 37 cents is for underground labor.

A report by J. B. Hadesty† shows that the sand filling system has not yet been adopted on a large scale in the western part of Europe,

*Gullachsen, Berent Conrad, "The Working of the Thick Coal Seams in Upper Silesia," *Trans. Inst. Min. Engrs.*, Vol. 42, p. 209, 1911.

†"Pennsylvania State Anthracite Mine Cave Commission Report," *Journal Pa. Legislature*, Appendix, 1913.

and the statements on cost of filling show that it would be impossible to adopt the process in the United States without materially increasing the cost of production.

It is unnecessary to go into the methods of mining and the results obtained in other coal producing districts. While there are great coal deposits in other parts of the world, and while large quantities of coal are produced, these regions have not been developed sufficiently to work out what may be called a settled practice. No other districts, moreover, are really large producers of coal in the same sense as those already considered. The problems to be considered in connection with districts only partially developed, or districts which though well developed supply only a limited market, are different from those in this country, and the results in such districts are no indication of what can be accomplished here.

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